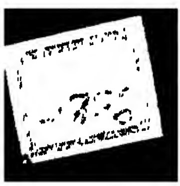

TRANSACTIONS

OF THE

American Institute of Mining and Metallurgical Engineers

(INCORPORATED)

Vol. LXXII

Containing Papers and Discussion on Mining and Sampling Presented at Meetings Held in New York in 1922, 1923, 1924, 1925, and offered for 1926; San Francisco, September, 1922; Birmingham, October, 1924; and Salt Lake City, September, 1925.

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PREFACE

Choice of the method for mining the ore from a particular deposit is often difficult. At present it rests on judgment and experience with all too little formal knowledge to guide the engineer. Especially for large irregular masses is it difficult and the success or failure of a mining enterprise may be determined by this decision, which necessarily must be made before mining begins. It is not that experience is lacking; many methods have been used in many lands and through sufficient time to determine their success under various conditions, but there is an absence of definite criteria and such formulation of terms as permit the easy transfer of knowledge from mine to mine or district to district. Even personal judgment and personal experience, carried from one field to another, are by no means always certain of success. Large as have been the benefits from the introduction of caving methods used in the iron mines of the Lake Superior into the copper mines of the West, the most competent and active engineers responsible for that advance admit freely that mistakes were made. Mining is such a complex process, so many things enter into it, that in the absence of a more complete analysis of the facts known, each new undertaking involves a risk in the choice and application of even an old and well-known method.

The Mining Methods Committee fully realizing the difficulties of meeting this situation resolved, about three years ago, to begin a systematic study of which this volume is the first fruit. It was thought that if a series of descriptions of methods in actual use, prepared in terms admitting a ready comparison, could be brought together there would be at least a basis for study. It was thought important to determine the efficiencies of the various methods of extraction in use as regards percentage of recovery, for there is a lack of data as to actual amount of ore now as compared with original estimates. In order also to meet the situation created by higher costs, it was desired to determine the efficiencies per man or per machine, under various conditions, with the hope that dissemination of knowledge as to the best practice would help to raise average practice. Unit costs rather than total costs were desired and the proportion of costs chargeable to labor, supplies, taxes, and other items was considered important.

A plan for a series of papers designed to meet this need was prepared and members of the Institute in various districts were requested to pre-

pare reports written according to it. There has been a generous, but still incomplete, response. Important districts and a number of special methods of mining are not represented in the series. In a number of instances, the authors either could not get or have not presented much of the data requested. Accordingly, the comparisons that can be made are disappointingly few. Such as seem permissible are, in the main, embodied in Table 1 and additional data will be found in the introductory statements of the various systems of mining.

It has been deemed advisable to publish all the papers that have been presented, and from these a table has been made up comparing results of the different methods that have been selected for the special conditions of ore and country rock. The results of extraction by any method and the per cent. of extraction have not been brought out in any of the papers, and it is hoped that attention will be given to these important points and the others covered by the original scheme in such later papers as may be offered for publication.

The subject of mine sampling and estimation of orebodies is receiving particular attention for this and the coming year, and in this connection possibly some of the missing data may be offered. The papers so far presented on this subject have been particularly interesting and valuable. The subject is mentioned in several of the more important papers aside from those especially devoted to it and a résumé has been prepared from the information given. This has been revised by Charles Janin in the absence from the country of I. A. Joralemon, Chairman of the Sub-Committee on Sampling.

The compilation of this volume has been done by Mr. M. W. von Bernewitz, under the supervision of Prof. R. M. Raymond, now Chairman of the Committee. The papers and discussions included are those presented at the Meetings held in New York in 1922, 1923, 1924, 1925, and offered for 1926, together with those read at San Francisco in 1922; Birmingham, 1924; and Salt Lake City, 1925.

Imperfect and incomplete as this survey of mining methods is recognized to be, it is presented in the hope that its study will stimulate research and lead to exactness in presentation of data of production and costs in place of the present indefiniteness.

H. Foster Bain,
Secretary

CONTENTS

Mining Methods Committee and Outline for Papers on Mining Methods

The membership of the Mining Methods Committee in 1922, when this survey of mining methods and of results was undertaken, is given below. Following is the outline prepared by the committee and distributed for the use of the members chosen to write the various papers. The membership of the subcommittee on coal and the special outline issued for use in that connection are given later in the volume.

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Chairmen of Special Committees

Classification of Mining Methods.—F. W. Sperr.
Anthracite.—R. V. Norris.
Bituminous Coal.—Howard N. Eavenson.
Copper.—William B. Daly, Northwest; Fred W. Denton, Lake Superior; Gerald F. Sherman, Southwest.
Iron.—Earl E. Hunner.
Lead and Zinc.—Arthur Thacher.
Precious and Rare Metals.—George A. Packard.
Sampling and Estimating.—Ira B. Joralemon.
Non-metallic Ores.—H. Ries.

AN OUTLINE FOR PAPERS ON MINING METHODS*

Foreword

The Mining Methods Committee during the past year has spent much time developing the interest of members of the Institute in the work that comes under its direction. The response on the part of the membership has been most encouraging and in order to give a concrete form to the program which it hopes to develop, an "Outline for Papers on Mining Methods" has been drawn up and is herewith submitted for consideration. This outline has slowly taken shape and embodies the ideas of many men; it is hoped that the outline will be suggestive and helpful in the preparation of papers. It can be used for the preparation of reports covering single large operations or, preferably, as the basis for a description of methods and the presentation of results obtained either in different districts or in different branches of the industry.

The Committee presents this outline for the consideration of the various Local Sections of the Institute with the request that they will endeavor to interest the members of the Sections in covering comprehensively the mining operations with which they are connected.

It is not the desire of the Committee to publish a paper on every mine, giving all the details outlined, but rather that an engineer familiar with a district prepare the general information and engineers of each important mine in the district compile the specific information of the mine with which they are connected. The general information and the specific information will be sent to the Committee and from this information the Committee will prepare a paper on the results obtained with the various mining methods in the different districts. The Committee requests that all papers be compiled under the direction of the chief engineer of the company and then submitted to the general manager before acceptance for publication or compilation and tabulation with other similar data. Where the study of this subject approaches the fields of other standing committees of the Institute, the Committee invites them to carry on the work for it or with it as they may desire.

The Committee, realizing the immensity of its field, proposes to divide its activities among a number of special committees, as follows:

Advisory Committtee	Copper
Classification of Mining Methods	Iron
Sampling and Estimating of Ores	Lead and Zinc
Anthracite	Precious and Rare Metals
Bituminous Coal	Non-metallic Ores

* Issued February, 1922.

On account of the size of the copper-mining field, three committees are provided for: one with headquarters in Butte, attending to vein deposits; a second, covering the Lake Superior District; the third, with headquarters in Bisbee, covering the so-called porphyries; and the other, massive deposits. The membership of these committees is given, and it is requested that all members interested communicate with the chairmen of these committees or with the offices of their Local Sections.

An outline of this kind is necessarily comprehensive, but it is not the thought that all the topics brought out should be covered in detail, but that the important points in each operation or district should receive special attention. It is earnestly hoped that the membership of the Institute will coöperate with the Mining Methods Committee to the end that the *Transactions* of the Institute may constitute a real record of progress in the industry.

Outline for Papers

To be followed as closely as practicable by the engineer in each district. Important features not covered by this outline should, of course, be brought out, giving proper weight, however, to the fact that the information desired is a full and detailed description of the principal mining methods employed in the district, with sufficient data and other information to bring out the reason for its adoption in preference to other operating methods. Paragraphs 1 to 6 inclusive, dealing with general information of a district, may be written by one engineer in each district. Data in the remainder of the outline may be compiled by another engineer or engineers from specific information obtained relative to the principal mining methods used in the district.

All of the information should be combined in one paper accompanied by detailed drawings and sketches necessary to show clearly all interesting details incident to exploration, development, and operation of a property.

1. Name and location of mining district.

2. History of district, briefly told, to date: (*a*) Give unit size of mineral tracts. (*b*) State whether ownerships are held in fee or lease, or both.

3. Geology of district, briefly told, giving age and kind of rocks, type and shapes of deposits, mineralogy and physical structure of orebodies and veins, amount and character of overburden. When important, bring out the fact that the mining method selected is determined largely by character of occurrence of minerals in the deposit.

4. Description, briefly told, of general items that will influence methods of mining, such as topography of district, climate, altitude, water and fuel supply, timber and other material supplies, character and efficiency of labor (native or alien, union or non-union) accessibility, methods of transportation, possible pumping problems.

5. Exploration, sampling, and estimating methods used to determine size and mineral contents of orebodies preliminary to mine development:
   (*a*) Exploration: churn or diamond drilling, test pitting, tunnelling, trenching, shaft sinking and drifting.
   (*b*) Sampling: character of ore, hardness, friability, moisture, methods of taking samples, methods of averaging.
   (*c*) Estimating: plans, cross-sections, cubic feet per ton, mining losses, remarks.
   (*d*) Accuracy of methods, as indicated by exploitation, to be stated if possible as percentage of production over or below estimate.

6. History of principal mining methods formerly in use, briefly told, and reasons for changing to present principal methods of district, such as the following: influence of legal restrictions or of lease conditions, if any exist; influence of the subsequent treatment or marketing of the product, etc.

7. Full discussion of principal mining methods of district. In districts where both underground and surface mining are practiced, show application of factors determining whether ore deposits should be mined by underground or open-pit methods. Under the following two subheadings give fully the more important details of the methods in use at the particular mines described, following the outline as nearly as practicable:
   (*a*) Underground mines, (*b*) open-pit mines.

Underground Mines

1. Briefly describe the more important features of the orebody and surrounding rocks that affect the operation of the mining method, such as the shape and size of orebody, its length, position, dip, structural conditions, hardness, grade or value, distribution of minerals through deposit and method of grading, occurrence of waste bodies in ore zone, character of wall rocks and capping or overburden as affecting support required and flow of ore, minimum angle of gravity flow of broken ore, influence of water on flow of ore, ratio of waste mined to ore, rock temperatures and influence of the same, and of any gases generated, on ventilating problem.

2. MINE OPENINGS, SHAFTS OR TUNNELS, size, depth or length, construction, timber, steel, concrete—reasons for location. Give sketches of shafts or tunnels with dimensions; also operating features determining type; also loading and discharging pockets. Show advantage of permanent location and construction in sinking deep shafts serving large orebodies.

3. UNDERGROUND DEVELOPMENT PLANS; main levels, sublevels, raises, chutes, etc. This may be shown by sketch giving plan of development on main and sublevels, etc., with relative dimensions and intervals; also gage, grade, radius curvature and weight of rails of track; method of conveying water on levels; size of compressed air and ventilating pipes.

4. MINING

(*a*) Drilling and blasting. Types of drills, shape and size of steel, form of drill bit; arrangement of holes drilled in drifts and stopes; method of tamping and firing holes, with comments relative to subsequent bulldozing at grizzles or chutes; kind and grade of explosives. Give special note if method involves breaking to minimize loss in fines.

(*b*) Drifting and stoping. Description of drifting and stoping methods, giving details of practice and arrangement of drifts, raises, sublevels, and other development work to the stope. State this in such manner that the tons extracted per foot of development work can be calculated. Furnish necessary sketches illustrating methods. For filled stopes give details of practice and explain how waste filling is obtained. State whether or not ore is sorted and, if so, whether in the stopes or above ground; describe any equipment used in this connection.

(*c*) Timbering. Describe kinds used in drifts, sublevels and stopes, preservatives used, method of delivery to working places, framed on surface or underground. Is any timber recovered for reuse?

(*d*) Underground sampling and check sampling of product on surface. Where taken, at what intervals, mechanical or by judgment? Method of cutting sample.

(*e*) Loading machines and scrapers.

(*f*) Tramming and haulage. Type and size of haulage motors: types of cars used, capacity, motive power; weights of rail as compared with car capacity, gage and grade of track.

(*g*) Underground storage and dumping. Pockets or large station area for cars; dumping direct to skip or through pocket; use of rotary dumps, automatic measuring pockets, etc.

(*h*) Hoisting. Type and size of hoists, cages and skips, capacities and hoisting speeds; safety catches on cages; size and type of

hoisting rope; special features of importance; kind of power; hoisting in balance or not.

(*i*) Pumping. Type and capacity of pumps; power.

(*j*) Air compression. Type and size of air compressors; power.

(*k*) Ventilation. Natural or artificial? Type, capacity, and location of fans; description of air distribution system, showing how air currents are made to reach the actual working faces; describe any equipment or special devices used in this connection.

(*l*) Lighting. System used for general lighting in shafts, stations, drifts, etc. System used for individual miners.

(*m*) Telephone. System used for general communication underground. Method of signaling in haulage and in shaft.

5. Records of unit production.

(*a*) Total labor cost expressed in percentage of total cost of mining.

(*b*) Contract or day labor; contracts based on car, ton, or foot unit.

(*c*) Tons per man per hour and also man-hours per ton for miners in stopes (all men in stope).

(*d*) Tons per man per hour and also man-hours per ton for miners working on development in ore.

(*e*) Tons per man per hour and also man-hours per ton for miners working on development in rock.

Note.—Possibly cubic yards would be a better unit to use in this case.

(*f*) Tons per man per hour and also man-hours per ton for all miners. (Take place of (*c*), (*d*), and (*e*) if these are not answered).

(*g*) Tons per man per hour and also man-hours per ton for all underground labor.

(*h*) Tons per man per hour and also man-hours per ton for surface labor (all working force on surface, exclusive of office).

(*i*) Tons per man per hour and also man-hours per ton for total organization including office force.

(*j*) Classification of labor, expressed in percentage of total.

(*k*) Labor turnover.

6. Records of units of supplies used, per ton of ore produced.

Note.—For purposes of this discussion only major items of supplies such as explosives, timber and power (fuel or electricity) have been itemized, and all other supplies taken together.

(*a*) Explosives, pounds per ton.

(*b*) Timber, feet B.M., linear feet or cubic feet per ton; lagging or poles, feet B.M., cords or cubic feet per ton; boards, B.M. per ton.

(*c*) Horsepower (total) per ton. Also divided into following subheadings: mining (compressed air) per ton; haulage and hoisting per ton; pumping per ton; ventilation per ton; lighting per ton; cubic feet compressed air used per ton.

(*d*) Cost of supplies used but not taken into account above, expressed in percentage of total cost of production.

(*e*) Total cost all supplies expressed in percentage of total cost of mining.

7. ORE MINED. When possible, give percentage of estimated ore that is actually mined.

8. DISPOSITION OF ORE AFTER REACHING SURFACE, to stockpile, mill or shipment.

9. SAFETY AND WELFARE WORK, briefly told.

NOTE.—All figures under 5 and 6 assume a fully developed mine operating under normal condition. Figures under 6 should assume that production costs cease with dumping of ore either into stockpile or shaft headframe pocket.

Open-pit Mines

1. Describe the more important features in connection with the deposit that affect the operation of the pit: topography in vicinity of deposit; depth and nature of overburden; size and depth of orebody; general structural conditions of orebody; grade or value of ore; distribution of values through deposit and methods of sampling and grading; occurrence of waste bodies in ore zone; character of wall rocks; safe operating slopes in various materials; alignment and grade of tracks; location of repair and supply shops and water and coal facilities for operating equipment; location and grade to lean ore and waste dumps; location and capacities of, and grades to, stock yards; other points that may have important bearing; furnish sketch showing typical deposit with pit plans and location of necessary tracks, buildings and waste dumps serving same.

2. STRIPPING.

(*a*) Height and width of cut; width of berm used under various depths of stripping.

(*b*) Machinery and equipment, giving number, type, size and capacity of each of the following used: Hydraulic giants; shovels; drag lines; locomotives; cars; locomotive cranes; track lifters; dump spreaders; tie tampers; drills; tracks, gage, weight required for various types and weights of equipment.

3. MINING ORE.

(*a*) Drilling, blasting, type of drill, shape and size of steel, frame of drill, arrangement of holes, method of firing holes, kind and grade of explosives.

(*b*) Methods of removal, locomotive haul, or mills and chutes with hoisting.
(*c*) Width and height of cut for different types of shovel equipment used in pit.
(*d*) Machinery and equipment, giving number, type, size, and capacity of each of the following used: Shovels; drag lines; locomotives; cars; locomotive cranes; track shifters; tie tampers; drills; hoisting facilities, tracks, gage, weight required for various types and weights of equipment.

4. Records of unit production, stripping operations.
(*a*) Period reported on.
(*b*) Total cubic yards stripped.
(*c*) Ratio of yardage stripped to tonnage made available.
(*d*) Cubic yards per man per hour and also man-hours per yard for stripping crews.
(*e*) Cubic yards per man per hour and also man-hours per yard for entire portion of organization charged to stripping.
(*f*) Average cubic yards per varying sizes and types of shovels per hour for full operating time, also based on actual results per hour during period operated.
(*g*) Length of haul from shovel to dump.
(*h*) Grade of track.
(*i*) Average number cars per train.
(*j*) Percentage of total time required for shoveling stripping.
(*k*) Percentage of total time required for hauling stripping.
(*l*) Percentage of total time lost (moving shovels, breakdowns, etc.).
(*m*) Total cost of all labor, expressed in percentage of total cost of mining.
(*j*), (*k*), and (*l*) should equal 100 per cent. of operating time.

5. Records of units of supplies used, per yard stripped.
(*a*) Explosives, pounds per cubic yard; give strength and character of explosives used.
(*b*) Fuel, pounds per cubic yard or express in terms of horsepower per cubic yard, if possible; where electrical power is used, express in terms of kilowatt-hours per cubic yard.
(*c*) Cost of repairs and maintenance, expressed in percentage of the total cost of stripping per yard.
(*d*) Cost of supplies not taken into consideration above, expressed in percentage of the total cost of stripping per cubic yard.
(*e*) Total cost of all supplies expressed in percentage of total cost of mining.

Note.—All figures under 4 and 5 should cover various kinds of material encountered in stripping, such as surface overburden, broken rock and solid rock where same occurs in such abundance as seriously to affect the cost of the operation.

6. RECORDS OF UNIT PRODUCTION, mining operations.
   (*a*) Total production for period reported on (specify long, short, or metric tons used).
   (*b*) Length of haul from shovels to stock yards.
   (*c*) Grade of track.
   (*d*) Average number cars per train.
   (*e*) Tons per man per hour and also man-hours per ton for pit crews.
   (*g*) Tons per man per hour and also man-hours per ton for entire portion of organization charged to production, including mine office.
   (*h*) Average tons per varying sizes of shovel per hour for full operating time, and also based on actual results per hour during period operated.
   (*i*) Percentage of total operating time required for shoveling ore.
   (*j*) Percentage of total time required for hauling ore (or hoisting).
   (*k*) Percentage of total time lost (moving shovel, breakdowns, etc.).
   (*l*) Classification of labor, expressed in percentages of total.
   (*m*) Labor turnover.
   (*n*) Total cost of labor, expressed in percentage of total cost of mining.

7. RECORDS OF UNITS OF SUPPLIES USED, per ton of ore produced.
   (*a*) Explosives, pounds per ton; give strength and character of explosives used.
   (*b*) Fuel, pounds per ton or express in terms of horsepower per ton, if possible; where electrical power is used, express in terms of kilowatt-hours per ton.
   (*c*) Cost of repairs and maintenance of equipment, expressed in percentage of the total cost of production per ton.
   (*d*) Cost of supplies used but not taken into account above, expressed in percentage of the total cost of production per ton.
   (*e*) Total cost of all supplies, expressed in percentage of total cost of mining.

8. ORE MINED.

   When possible, give percentage of estimated ore that is actually mined.

9. SAFETY AND WELFARE WORK.

NOTE.—All figures under 4, 5, 6, and 7 assume a fully developed mine operating under normal conditions. Figures should assume that stripping costs cease when waste material is deposited on waste dumps and ore costs cease when trainloads of ore from pit are deposited in railroad receiving yards at approach to pit. It is assumed that all figures called for cover a period when the operation is conducted under normal conditions.

Classification of Mining Methods

A special committee, after careful consideration and discussion, decided on the following classification as being sufficiently distinct and most convenient for use in the preparation of papers on Mining Methods.

- **Open cut, with or without stripping**
  - Benches
  - Glory holes and milling
  - Placer mining
    - Hand sluicing
    - Dredging
    - Hydraulicking
- **Underground methods**
  - *Breast stoping* (tabular, flat-dipping deposits or beds)
    - Continuous horizon
    - Casual pillars
    - Room-and-pillar
  - *Underhand stoping* (veins and large masses)
    - Glory hole and milling
    - Open stope
      - With pillars
      - With stulls
      - With square-set timbering
  - *Overhand stoping* (steep-dipping veins or masses
    - Open stope
      - Will pillars
      - With stulls
      - With square-set timbering
    - Filled stope
      - Horizontal cut and fill (flat-back)
      - Inclined cut and fill (rill)
      - Square-set timbering
      - Resuing or stripping
    - Shrinkage stope
    - Sublevel stopes
  - *Top slicing* (wide veins or masses)
    - Continuous horizon
    - Panel slicing
    - Block slicing
    - Inclined slicing
  - *Caving* (large masses)
    - Sublevel caving
    - Block-caving
      - Undercut from main level
      - Undercut from sub-levels
      - Back caving into chutes
      - With branch raises
- *Combined Methods* (large masses)
  - Alternate shrinkage stope, and pillar Pillars mined by: top-slicing, Sublevel caving or block-caving
  - Square-set timbering: pillars mined by slicing

Result of Mining Methods Committee's Efforts

The response to the request for information as outlined was the 46 papers published in this volume. Of these papers, 29 relate to metal mining, 7 to coal mining, and 10 discuss sampling and estimation of tonnage. The volume has been divided into three parts: Metal-Mining Methods, Sampling and Estimating Ore Deposits, Coal-Mining Methods.

Each part has been subdivided according to metals, and the discussion of sampling and estimating is taken from the information given in the mining papers. Naturally, the methods and costs of metal mining and sampling have received the most attention, and the papers on coal mining, though few, are excellent examples of advanced practice.

The important part of the information that was desired is the production by the method, or methods, best adapted to the character of the ore, and the proportionate expense in labor and supplies for underground operations. Table I is an interesting comparison of the different methods and includes, in order to cover the larger field, similar data from papers published in the *Transactions* as far back as 1904. In the earlier papers there is a paucity of the production data but what is available is given.

In order that the better comparison of costs may be made, the purchasing price of the dollar, in terms of the wholesale price index number, based on 100 for the year 1913, is given. As one mine or district sometimes uses a number of methods, each is given separately in the table.

TABLE 1.—*Summary of Mining Methods and Costs*

Name and District	Metal	Country Rock	Ore	Hard or Soft	Width, Feet	Daily Output Tons[1]	Production per Man-day[2] Tons: Miners	Production per Man-day[2] Tons: All Underground	Labor Cost, Per Cent.: Breaking	Labor: Timbering	Labor: Shoveling	Labor: Tramming	Labor: Sundries	Labor: Total	Supplies, Per Cent.: Explosives	Supplies: Timber	Supplies: Air	Supplies: Power	Supplies: Sundries	Supplies: Total	Total Cost of Mining Ore, per Ton	Wholesale Price[3] Index Number: Year[4]	Wholesale Price Index: Number
Glory hole																							
United Verde,[5] Ariz	Copper	Quartz-porphyry and schist	Massive sulfides		Up to 200	450	26 1	25 5	21.4	3.65	2.97	23.2	9.3	60.6	16.8	1.3	7.7	0.34		39.4	$0.88	1918	194
Fresnillo,[5] Mexico	Silver-gold	Graywacke and slate		Soft	3 to 40	3,500	20 0	12 8	29.5			17.0	0.6	48.7	23.0			16.0		51.3	0.20	1923	154
Climax, Colorado	Molybdenum	Alaskite	Sulfide in quartz			300																1918	194
Mascot,[5] Tennessee	Zinc	Dolomite	Sphalerite in dolomitic limestone	Tough		1,400		8 5													0 65	1924	150
Caving																							
Inspiration,[6] Ariz	Copper	Schist	Chalcocite in schist and quartz porphyry			16,700																1914	98
Miami,[5] Ariz. . . .	Copper	Schist and granite	Chalcocite in schist and granite	Easily broken	1,600 (max)	6,000	10 8	7.6	50 9	12 7				63.6	7.3	23.6					$0 55	1916	127
Ray Cons.,[7] Ariz . . .	Copper																					1915	101
Alaska Juneau,[5] Alaska .	Gold	Slate and metagabbro	Gold and sulfides in quartz stringers	Friable		10,000 (approx.)	16 5	4.6	30.8			27.5		58.1	24.5			4.7	12.3	41.6	0.23	1923	154
Ruth,[8] Nevada.	Copper	Porphyry	Chalcocite in porphyry	Soft																		1917	177
Top-slicing																							
Phelps Dodge—Morenci Branch,[9] Ariz.	Copper	Quartz-porphyry granite and limestone			20 to 40	520		10 5 (all mine workers)						60.0						40.0	$0.65	1917	177

Calumet & Arizona,[16] Ariz.	Copper													65.1	4.5	21.2	4.5		3.0	33.2	0.93	1915	101
Marquette,[5] Michigan ...	Iron	Slate or diorite and jasper		Soft	Up to 1,200																	1922	149
Newport,[11] Mich.	Iron	Quartzite and slate	Hematite in jasper and quartzite	Soft		3,500																1911	93
El Bordo,[5] Mexico	Silver-gold	Andesite	Sulfides in quartz and altered andesite	Hard to loose	13 to 36	31,023 (total)		3.14						57.7	8.9	32.1			1.3	42.3	0.78	1925	160
Sub-level caving																							
Phelps Dodge—Morenci Branch,[8] Ariz.	Copper	Quartz-porphyry granite and limestone																				1914	98
Oceanic,[12] Calif	Quicksilver	Serpentine and sand rock	Cinnabar in mud rock	Soft	20																	1914	98
Mascot,[5] Tenn.....	Zinc	Dolomite	Sphalerite in dolomitic limestone	Tough		1,300		5.7													$0.70	1917	177
Open stopes with pillar supports																							
Phelps Dodge—Morenci Branch,[8] Ariz.	Copper	Quartz-porphyry, granite and limestone		Hard																	$	1914	98
Beatson,[5] Alaska.........	Copper	Graywacke and slate	Chalcopyrite in quartz	Easy to break	Up to 280	1,160		14.0						59.1						40.9		1922	149
Elko Prince,[13] Nevada....	Gold	Rhyolite	Sulfides and quartz	Hard	15	59			25.9	17.9		8.9			14.4		4 1				3.12	1917	177
Liberty Bell,[5] Colorado ..	Gold and silver		Sulfides and quartz and calcite	Easy to break	3 to 4	988,325 (total)		2.4						66 0				3.3	30.7	34.0	1.80	1922	149
Cornucopia,[5] Oregon	Silver and gold	Granodiorite	Sulfides and quartz		5 to 20	100	16.9	3.4													1.94	1923	154
Lucky Tiger,[5] Mexico.....	Silver-gold	Rhyolite	Sulfides in kaolinized rhyolite	Hard and soft	19	200		0.64	14.4	13 4	7.7	7.3	2.4	68 3	6 6			5.8	19.2	32.7	7 67	1924	150
Birmingham,[5] Alabama...	Iron	Shale and sandstone	Hematite		15 to 22																	1924	150

TABLE 1. *Summary of Mining Methods and Costs.*—(*Continued*)

Name and District	Metal	Country Rock	Ore	Hard or Soft	Width, Feet	Daily Output, Tons[1]	Production Per Man-day[2] Tons		Labor Cost, Per Cent.						Supplies, Cost, Per Cent.						Total Cost of Mining Ore per Ton	Wholesale Price[3] Index Number	
							Miners	All Underground	Breaking	Timbering	Shoveling	Tramming	Sundries	Total	Explosives	Timber	Air	Power	Sundries	Total		Year[4]	Number
Mineville,[5] New York	Iron	Gneiss	Magnetite and silica	Hard	4 to 70		36.8	5.9						58.0	13.0	1.9		10.3	16.8	42.0		1922	149
Missouri,[14] Southeastern..	Lead	Limestone	Galena and limestone	Tough				5.7													0.85	1911	93
Broken Hill,[15] Australia...	Lead-zinc silver	Schist-gneiss	Sulfides in rhodonite calcite and other gangue	Hard	Up to 350																3.60 to 4.80	1914	98
Winsconsin,[16]...........	Zinc-lead	Limestone	Sphalerite																			1917	177
						Timbered stopes																	
Phelps Dodge—Morenci Branch,[8] Ariz.	Copper	Quartz-porphyry granite and limestone		Hard																	$	1914	98
Butte district,[5] Montana	Copper	Granite or aplite	Sulfides in quartz and granite		10 to 30; range 4 to 100		5.0	2.0						68.0						19.4		1922	149
Calumet & Arizona,[9] Arizona.	Copper													61.2	5.0	23.7	5.0		5.0	38.7	0.80	1915	101
Moctezuma,[17] Mexico.....	Copper	Latite and andesite breccia	Sulfides		3 to 40	2,000	4.1							41.8	6.2	40.6	11.4			58.2		1924	150
United Verde,[5] Arizona...	Copper	Quartz-porphyry and schist	Massive sulfides		Up to 200	227			11.9	17.5	14.3	6.1	10.9	65.4	2.8	18.4		4.4		34.6	3.33	1918	194
Mother Lode,[5] California (costs of Plymouth mine).	Gold	Slate and schist	Free gold and sulfides in schist	Friable	10; up to 100	250	3.9	2.3						58.2	4.0	10.0		7.2	11.0	32.2		1922	149
Santa Gertrudis,[18] Mexico.	Silver-gold	Andesite	Sulfides and quartz	Hard	Up to 50 and 100	1,000															2.26	1916	127
Bunker Hill & Sullivan,[5] Idaho.	Lead-silver	Quartzite	Galena in siderite and quartz		40 to 125	1,150	4.2	3.6						43.5						19.4	2.80	1922	149
Hecla,[5] Idaho...........	Lead-silver	Quartzite	Galena in quartz		3 to 40	730	5.7	3.6													3.59	1921	147

Morning,[5] Idaho	Lead-silver	Quartzite	Sulfides in siderite and quartz		Up to 50		5.1							67.0						33 0		1923	154
Bawdwin,[19] Burma	Lead-zinc silver	Rhyolite and tuff	Sulfides	Hard to loose	50 to 140	560	1.6	0.54	28.1					57.0	8.0	17.0			18 0	43.0	3 07	1923	154
Broken Hill,[15] Australia	Lead-zinc silver	Schist-gneiss	Sulfides in rhodonite or calcite and other gangue	Hard	Up to 350																3 60 to 4.80	1914	98
Park City,[20] Utah	Lead-zinc silver	Quartzite and lime-stone	Oxides and sulfides in silicious gangue	Hard but breaks easily	10									32 7	6.2	35 8					1 91	1911	93
Filled stopes																							
Phelps Dodge—Moreni Branch,[8] Ariz.	Copper	Quartz-porphyry, granite, and limestone		Hard																	$	1914	98
Butte District,[5] Montana	Copper	Granite or aplite	Sulfides in quartz and granite		10 to 30, but range 4 to 100		6.4	2.0						68 0						19 4		1922	149
Calumet & Arizona[9]	Copper						12.0							66 6	7 8	13 7	7.8		3 9	33 2	0.51	1915	101
Copper Range,[5] Michigan	Copper	Conglomer-ate or sandstone	Native copper	Medium hard	16 to 26; up to 90	1,770	19.5	2.5						54 6						31.4	2.96	1921	147
Iron Cap,[5] Arizona	Copper	Schist, shale and quartz-ite	Sulfides in brecciated shale or quartzite	Hard	3 to 40	280	10 0	3.3						41.4	8.3	7.6	14.1	8.9	14 5	58.6	2.90	1923	154
Moctezuma,[17] Mexico	Copper	Latite and andesite breccia	Sulfides		3 to 40	2,000	4.7							54.7	11 2	14.8	19.3			45.3		1924	150
United Verde,[5] Arizona	Copper	Quartz-porphyry and schist	Massive sulfides		Up to 200	1,560			15 3	7 7	17.8	8.3		71.2	6 1	7.3	5 1	0 16	9.9	28 8	2 46	1918	191
United Verde Extension,[21]	Copper	Schist	Sulfides																			1918	194
Smuggler Union,[5] Colorado.	Gold-silver		Sulfides in quartz	Hard	2 to 2½		8 0	2 7						60 9				5.9	33 2	39.1	1.80	1922	149
Zaruma,[5] Ecuador	Gold	Greenstone	Quartz and calcite	Hard and tough	3 to 19	225		0 5													4.15	1920–'22	226–149
Lucky Tiger,[5] Mexico	Silver-gold	Rhyolite	Sulfides in kaolinized or silicified rhyolite	Hard and soft	19 in., stoping 39 in.	200		5 1	14 4	13.4	7.7	7.3	2 4	68 3	6.6			5 8	19 2	32.7	7 67	1924	150
Santa Gertrudis,[18] Mexico	Silver-gold	Andesite	Sulfides in quartz	Hard		1,000															2 26	1916	127

TABLE 1.—*Summary of Mining Methods and Costs. (Continued)*

Name and District	Metal	Country Rock	Ore	Hard or Soft	Width, Feet	Daily Output Tons[1]	Production Per Man-day[2] Tons		Labor Cost, Per Cent.						Supplies, Per Cent.						Total Cost of Mining Ore per Ton	Wholesale Price[3] Index Number	
							Miners	All Underground	Breaking	Timbering	Shoveling	Tramming	Sundries	Total	Explosives	Timber	Air	Power	Sundries	Total		Year[4]	Number
Broken Hill,[15] Australia...	Lead-zinc silver	Schist	Sulfides in rhodonite and quartz	Hard	Up to 350																3.60 to 4.80	1914	98
Silver King Coalition,[5] Utah.	Lead-zinc silver	Limestone and quartzite		Hard and soft			4.1	1.7						68.0						32 0		1922	149
Shrinkage stopes																							
Phelps Dodge—Morenci Branch,[8] Ariz.	Copper	Quartz-porphyry, granite, and limestone																			$	1914	98
Braden,[21] Chile..........	Copper	Andesite and tuff	Brecciated diorite and sulfides	Moderately hard	150			12.0				4.9									0.41	1909	97
Kennecott,[5] Alaska.......	Copper	Dolomite limestone	Chalcocite		Up to 100	990	11.2	3.7						55.2						44.8		1922	149
Moctezuma,[17] Mexico.....	Copper	Latite and andesite breccia	Sulfides		3 to 40	2,000	7.8							45.1	25.0	2.4	27.5			54.9		1924	150
Ray Cons.,[7] Arizona......	Copper	Schist and porphyry	Sulfides in schist																			1915	101
United Verde,[5] Arizona...	Copper	Quartz-porphyry and schist	Massive sulfides		Up to 200	390			32.5	0.36	0.68	9.3	12.3	55.2	11.8	3.6	11.8	0.2	17.2	44.8	2 19	1918	194
Utah Copper,[22] Utah.....	Copper	Monzonite	Mineralized porphyry			3,640		10.9						48 8							0.57	1911-'13	93-100
Alaska Gastineau,[23] Alaska.	Gold	Slate and greenstone	Gold in quartz			4,460															0.48	1915-'18	101-194
Alaska Treadwell,[24]......	Gold	Slate and meta-gabbro	Quartz and calcite			2,280			19.5			3.1		57 1	23.3			6.9		42.9	0.34	1902	84
Drumlummon,[25] Mont....	Gold	Hornstone and diorite			Up to 25																2 25	1887	81
Elkoro,[5] Nevada.........	Gold	Rhyolite	Quartz and adularia	Soft	3 to 10		15.1	3.4						51.3						30.2		1924	150

Homestake,[5] South Dakota.	Gold	Altered dolomite	Gold, sulfides, quartz and hornblende	Hard and tough		5,500	20.0	6.0						62.0					30.0		1923	154
Smuggler-Union,[5] Colorado.	Gold		Sulfides and quartz	Easy to break	14 to 24 (in.)	180	8.0	2.2						61.0			4.7	34.0	39.0	1.80	1922	145
Portland,[5] Colorado......	Gold	Breccia or granite		Hard	3 to 10	570	19.2	7.0						61.0					39.0		1922	149
Santa Gertrudis,[18] Mexico	Silver-gold	Andesite	Sulfides and quartz	Hard		1,000														2.26	1916	127
Mogollon,[5] New Mexico...	Silver-gold	Andesite and rhyolite	Sulfides and quartz and calcite	Hard	2½ to 24	160	10.8	1.8			23.0			61.0			12.3	26.6	39.0	2.75	1922	149

[1] If total for year was reported, tonnage found by dividing by 300 days, short tons, excepting iron ore.
[2] Tons per man-hour times 8 hours.
[3] U. S. Bureau of Labor Statistics. The price index number is weighted from 1890 and the year 1913 is the basis of 100.
[4] Period covered by papers cited.
[5] See paper in this volume (72); United Verde costs are from *Trans.* (1921) **66.**
[6] *Trans.* (1917) **55,** 218.
[7] *Trans.* (1915) **52,** 381.
[8] *Trans* (1918) **59,** 299.
[9] *Trans.* (1915) **51,** 267.
[10] *Trans.* (1917) **55,** 118.
[11] *Trans.* (1911) **42,** 676.
[12] *Trans.* (1915) **51,** 110.
[13] *Trans.* (1919) **60,** 78.
[14] *Trans.* (1914) **48,** 33.
[15] *Trans.* (1916) **54,** 20.
[16] *Trans.* (1918) **59,** 117; (1920) **63,** 213.
[17] *Trans.* (1912) **43,** 662; (1920) **63,** 382.
[18] *Trans.* (1917) **55,** 397.
[19] *Trans.* (1923) **69,** 208.
[20] *Trans.* (1911) **42,** 470; (1915) **51,** 281.
[21] *Trans.* (1919) **61,** 188.
[22] *Trans.* (1909) **40,** 743.
[23] *Trans.* (1916) **54,** 80.
[24] *Trans.* (1904) **34,** 334.
[25] *Trans.* (1914) **49,** 258.

Discussion of Data in Table 1

Selection of a Mining Method

The method to be adopted for extraction of ore depends on the character of the ore and the nature of its occurrence. The former Arizona Copper Co. had five orebodies that varied in width from 6 to 200 ft. and eight methods had to be used—open cut, open stope and filling, underhand stoping, shrinkage stoping, square-set and fill, sublevel caving, rill stoping, and panel slicing. Sometimes choosing a method is a case of deciding between a method that gives a high extraction of ore at a high cost and one that gives a lower extraction at lower cost.

Glory-hole Mining

The United Verde gives an example of glory-hole work as practiced in oxide ore and is to be used in conjunction with steam shovels to the 200-ft. level. The comparative cost will largely determine the method to be used in mining as far down as the 400-ft. level. At Fresnillo, where the output per man-day is 23 per cent. less than at Jerome, the cost is 75 per cent. less; of course, the scale of operation is much greater at Fresnillo, where the relative values of steam shovels and glory holes were carefully compared before the latter system was adopted.

At Mascot, shrinkage stoping was tried and abandoned because of the tendency of rock to break into slabs too large to pass through the chutes. The system now used is mill-hole mining, although on account of the relation of some of the mining levels to the orebodies it is necessary to work a number of open stopes. Ninety per cent. of all mining is done on contract and 52 per cent. by the mill-hole method. Under the latter system, the output has been increased from 4.45 to 8.50 tons per man-day. The cost of ore on cars underground is 24.1 cents per ton by the mill-hole system and 30.7 cents by open stoping, in some parts of the mine. The total cost of underground mining, less development, is 65.1 cents per ton.

Caving Methods

Four examples of caving are given and large tonnages are mined thereby. In the two cases where data are available, the quantities mined and costs vary from 23 to 55 cents per ton, although the percentage of labor costs is practically similar.

At Inspiration Consolidated, caving consists in the main of undercutting the ore (taking out a horizontal slice), allowing the ore to cave and crush, and drawing off the crushed ore through small inclined raises driven under the crushed ore into main inclined raises that lead down to the haulage drift.

At Ray Consolidated, a block of ore is weakened by a series of shrinkage stopes, or "ore-filled rooms." After undermining and shattering the

pillars remaining, all the broken ore is drawn systematically, the capping gradually crushing and settling upon it.

At Miami, undercut caving for pillars between shrinkage stopes superseded top slicing. More ore can be mined from a given area, fewer men are required, production is more elastic and easily controlled, ventilation is better, the fire risk is lessened, the hazard is less, and the profit is greater by caving than by slicing.

At Alaska Juneau, where the ore zones consist of slate or slate with metagabbro intrusions, together with gold-bearing quartz masses, stringers, and gash veins, a system of block caving and bulldozing in chambers has been adopted successfully at a cost of 23 cents per ton. Before the rock is milled, about 50 per cent. is sorted out as waste.

In this connection, it is interesting to mention that up to Jan. 1, 1925, the Inspiration, Miami, and Ray mines had produced a total of 95,979,349 tons of ore by variations of undercut-caving.[1] Each mine has developed a practice more or less distinctive and therefore differing in important details. Undercut-caving at present accounts for a daily production of 40,000 tons of copper ore and is, therefore, one of the most important mining methods applicable to large low-grade copper deposits. Costs range from 68 cents to $1.30 per ton.

Top-slicing

Three copper mines, two iron mines, and one silver-gold mine give illustrations of the top-slicing method. All produce large tonnages of ore and the cost ranges from 68 to 93 cents per ton. The percentage labor cost and supplies also are fairly similar.

Arizona Copper Co. used the top-slicing principally in the extraction of large and soft deposits. It is an application of longwall retreating worked under an artificial roof of timber called a "mat." Sublevel caving was sometimes employed in mining the upper part of an orebody preparatory to top-slicing. Top-slicing is used for most of the soft iron ore on the Marquette Range, and losses and dilution of ore are largely avoided.

At Bisbee, the Calumet & Arizona mine, in 1914, introduced the top-slice caving system to a large body of oxide ore in which heavy and swelling ground made the cost of square-set stoping excessively high. While the method was partly successful and reduced the cost of stoping, M. W. Mitchell eventually evolved an inclined top-slice caving method which reduces the large amount of handling of ore in the stope, incidental to ordinary top-slicing, to a minimum. The "Mitchell" top-slicing method, as distinguished from the Mitchell slicing method, is applicable to the same conditions as those under which ordinary top-slicing is successful, which is uniform ground too heavy for economical square-set

[1] *Eng. & Min. Jnl.-Pr.* (1925) **120**, 243

stoping and which will cave readily as required, but may be at all times kept under adequate control. It may often be used with increased economy in ground adapted to square-set stoping if the other conditions obtain. The cost in oxide ore at Calumet & Arizona was 93 cents per ton in 1916.

At El Bordo, Pachuca, Mexico, the conditions mentioned given by R. J. Mechin show a useful adaptation of this method. There are three classes of ore mined—ore in place, old pillars and caved material, and old filling. Square-set and filling methods were employed at Santa Gertrudis but it was slow and costly, so top-slicing was adopted at El Bordo. The cost is 78 cents per ton, which is close to the other figures given. One pound of 40 per cent. dynamite broke 4.35 tons of ore, and 7.1 ft. board measure of square timber and 4.5 ft. of round timber were used per ton of ore.

Sublevel Caving

Sublevel caving was sometimes employed by the Arizona Copper Co. in the upper part of an orebody preparatory to top-slicing. Another case is that of the Oceanic quicksilver mine where sublevel mining proved to be the best and cheapest method that could be practiced with safety. In the irregular zinc deposits at Mascot, this method is satisfactory for a portion of the operations and costs 70 cents per ton mined.

Open Stopes

While there is considerable variation in the total cost of mining by the open-stope method (85 cents to $7.67 per ton) the ratio of cost of labor to supplies is approximately the same. In general, the rock enclosing the veins worked by open stopes is hard and the veins well defined, although in some cases the deposits are irregular. Such deposits as in the Marquette Range (hard ore), in southeastern Missouri, and in Wisconsin are mined by breast stoping. Pillars are generally left, as at Beatson and in the Birmingham and Mineville districts, and extracted later. In the Arizona Copper Co.'s mines, open stoping without timber was used with subsequent filling. At Cornucopia, Liberty Bell, and Elko Prince, the stopes, nearly vertical, are stulled and no attempt is made to fill them. At El Tigre, open stopes are used when the ore extends only short distances (20 to 30 ft.) above or below a level, and the walls are firm.

Timbered Stopes

In general, and naturally in soft and decomposed ground, the cost of timbering stopes is relatively high, although in the Calumet & Arizona the total cost of mining is only $1.04 per ton of ore mined. There it has been found that the square-set system is easily the most flexible; and where the orebody is irregular and has large included blocks of waste, this

system is the most satisfactory. It is simple to leave the waste behind as part of the filling, and where the mining of too large a section at once is not attempted, and filling is kept within a reasonable distance of the back, the square-set system is as safe as, or safer than, any of the methods in vogue. By this method the cost of stoping ranges from 80 cents in sulfide ore to $1.30 per ton in oxidized ore.

The Arizona Copper Co. found square sets too expensive in its large orebodies and substituted top-slicing methods, although square sets are used in mining orebodies of irregular shape and in the first stage of top-slicing. An excessive quantity of timber had to be used to reinforce the square sets to prevent serious caves.

At Butte, 80 per cent. of the ore is extracted by square-set mining. Shrinkage stoping was never a success because of the character of the walls.

At United Verde, only 6 per cent. of the ore is now extracted by square-set mining, and only in ground not strong enough for the cut-and fill method; also in coming to the top of the filled stopes. Square sets are used entirely in heavy ground where secondary enrichment occurs in the United Verde Extension.

At Moctezuma, 22 per cent. of the ore is extracted by the square-set method. It is used in taking out pillars between old cut-and-fill sections and between old caved shrinkage sections; also in removing old caved sections in the upper levels.

As the ground of the Mother Lode of California swells considerably, square sets and filling is the principal system of mining. So it is at Bunker Hill & Sullivan where the orebodies are wide and irregular and have a heavy hanging wall; also at the Morning mine in the same district. At the Hecla, also adjacent to the two last-mentioned mines, where there is necessity for close filling to give protection against side weight, filling and timbering are done practically at the same time.

At Pachuca the remining of old areas where old fills are now payable, and also where profitable ore in place is found on the walls, and at Santa Gertrudis a system of overhead stoping with timbering and filling is employed; this method is being supplanted by top-slicing, as at El Bordo and described under top-slicing. At Park City, Broken Hill, and Bawdwin square sets and filling have been found to be the best system.

At Broken Hill, from 5 to 100 per cent. of the ore is mined by the square-set method, and in the largest mines generally from 95 to 100 per cent. Where there is sufficient strength in the orebody to maintain an open stope, the whole orebody is removed by square sets, which are completely filled with mill tailings.

In general, little timber is recovered from square-set stopes; there is considerable risk and expense in such recovery. Calumet & Arizona sometimes recovers 50 per cent., which is used in other stopes. In other

districts the little timber that is recovered is more or less rotted and broken and cannot be used for regular timbering.

Filled Stopes

Except two low-cost mines and one high-cost, because of their exceptional conditions, the cost of mining by the filled-stope method ranges from $1.80 to $4.80 per ton; although in several places costs are practically the same and the percentage cost of labor and supplies are much the same as well as the tonnage per man-hour.

Where the ore is hard and stands well at Metcalf, Ariz., a system of open stoping without timber is used with subsequent filling. Waste filling follows behind the working face. At Calumet & Arizona, the Gilman cut-and-fill method, a special system, reduced costs to an average of 50 cents a ton and gave an output of 12 tons per man-day in 1916. At Houghton, the Copper Range uses both waste rock sorted out from the lode and mill sands for filling. At Iron Cap, the filling follows closely behind ore stoping. At Moctezuma, 50 per cent. of the mining is by flat cut-and-fill method, but it is being superseded by shrinkage stoping. At United Verde, 66 per cent. of the ore is mined by filled-stope methods. All waste filling comes from the surface. At Zaruma, filled-rill stopes superseded shrinkage stoping because the walls are unsuitable for shrinkage. Costs here are higher by the filling method, but as much as 40 per cent. more ore is obtained from a given orebody. Cut-and-fill methods are most commonly used at El Tigre, and are especially suitable where the veins are narrow (24 in. or less) and the walls soft. Filling follows mining closely. Similar procedure obtains at Telluride on the Smuggler vein. The filled rill is the second most important mining method at Butte, and is replacing to some degree the square-set method. At Silver King Coalition, about half of the stopes are filled—in other words, it depends on the strength of the walls whether the stope is filled or not.

The Mitchell slicing system, as distinguished from the Mitchell top-slicing method, is included under the head of "Filled Stopes." It is a combination of square-setting, mining pillars, and filling with waste, thus keeping the overburden intact. It is applicable to orebodies in which the hanging wall is fairly flat and regular and the lateral pressure not too great. The Mitchell slicing system is extensively used in the Calumet & Arizona mines and is termed Mitchell stoping in the United Verde Extension. At Bisbee, the actual saving in stoping cost at Calumet & Arizona is from 20 to 30 per cent. of that of square-set stoping. The cost in 1916 ranged from 85 cents per ton, for oxide ore, to 66 cents, for sulfide ore. The ore is rapidly mined and the output per man is much increased, sometimes three or four times that by square-set mining.

Shrinkage Stopes

More data is available on this method than the others. Omitting the costs reported several years ago, the present cost ranges from $1.80 to $2.75 per ton. Labor charges range from 51.3 to 62.0 per cent. and supplies from 30.0 to 44.8 per cent. of the total mining cost. The output per man, for all underground men and on the basis of an 8-hr. shift, is from 1.8 to 7.0 tons.

At Kennecott, the character of the ground makes almost an ideal condition for this method. No filling is needed, as a rule, but waste rock from development is dumped in the stopes after ore is drawn.

At Moctezuma, 26 per cent. of the ore is mined by shrinkage stopes, and this is being increased by supplanting the flat cut-and-fill method. Strong walls are essential when drawing ore.

At the Homestake, where stopes are opened across the orebody, the ore is all drawn out after being blasted and the space is filled with waste rock. The ore is hard and tough and stands well, as does the wall rock.

At Jarbidge, where the veins range from 3 to 10 ft. in width, shrinkage stoping is the only method employed. About 23 per cent. of the ore is drawn from the stope during stoping operations. Stopes are carried up as high as 250 ft. with little timber.

At Mogollon, the wall rocks of rhyolite and andesite stand well without timbering. The veins are from 8 to 15 ft. wide and shrinkage stoping has been found to be the best and cheapest method. This is the case also at Telluride and Cripple Creek.

Combined shrinkage with pillars and pillar caving are practiced at Braden, Moctezuma, and Ray, and were used at Alaska Gastineau and Alaska Treadwell at low cost. In fact, five mines in 1902, 1909, 1911, 1915, and 1924 reported low costs for shrinkage stoping—84 cents. 41 cents, 57 cents, 48 cents, and 83 cents per ton, respectively.

METAL-MINING METHODS

Glory-hole Mining

GLORY-HOLE MINING, also known as underground milling, is an adaptation of open-cut methods. The method is limited to large ore-bodies, masses, or wide veins with strong walls and ore. The top of a raise is widened in all directions, which results in a funnel-shaped opening (glory hole or mill hole), which is widened and deepened by underhand stoping. The ore is "milled" into the raise and is loaded through chute gates into cars on a level beneath. A large tonnage is mined by glory-hole methods, although less than formerly. Two good examples are described in this volume; namely, Fresnillo (silver gold) and Mascot (zinc).

Glory-hole Mining at Fresnillo*

By Thomas C. Baker†

The Fresnillo unit of the Mexican Corporation, S. A., is situated at the old historic mining town of Fresnillo, Zacatecas, Mexico, 33 miles north of the city of Zacatecas and 750 miles south of El Paso, Tex. It is connected with the Mexican Central Railroad by a broad-gage spur 5.5 miles long, belonging to the company.

Fresnillo is surrounded by a broad semi-arid valley on the Mexican plateau at an elevation of 7300 ft. The monotony of the valley is broken by a number of low hills. Near the town of Fresnillo is the "Cerro de Proaño," an elliptic-shaped hill of gentle slope, rising out of the plain to a height of 325 ft.; the base has a major axis of 4500 ft. and a minor axis of 3000 ft. In this hill is the silver-bearing mineralized zone, which comprises past and present mining operations. (In this paper money values are expressed throughout in American currency and the dry short ton of 2000 lb. is used.)

The discovery of ore at Fresnillo is credited to the Spaniards, soon after the conquest of Mexico. There are, however, no authentic records to show the amount of work done by them. In 1832, the State of Zacatecas took possession of the property and operated it. Evidently the rich ore above the horizon of the plain had been fairly well exhausted, as an old record states that thirty shafts were used for unwatering, requiring 300 men and 1500 animals. The cost of pumping alone amounted to $150,000 a year. In spite of the fact that cheap convict labor was used, mining costs were reported as $20 and treatment charges as $20 per ton.

In 1835, the state was obliged to seek foreign capital, and a company formed in London secured a controlling interest in the property. This new organization sank two shafts, one 11 by 19 ft. and the other 10 by 15 ft., to unwater the mines and extract ore. Large Cornish pumps were purchased in England and hauled overland from Vera Cruz; engines also were installed to operate the arrastres (grinding mills) in the patio process of ore treatment. Substantial buildings were erected and operations were conducted on a large scale, for that period. The patio, the largest in Mexico, was 1150 ft. square. The costs for 1850 were:

* Reprinted by arrangement with *Engineering and Mining Journal-Press* (Dec. 1, 1923) **116**; and presented at New York Meeting, 1924.

† Assistant to General Manager, The Mexican Corporation.

	Per Ton.
Mining	$ 4.86
Pumping	1 35
General	1 28
Treatment	4.64
Total	$12.13

From 1857 to 1867, Mexico was torn by civil strife, followed by the French occupation and unsettled conditions. During this period the water was allowed to rise in the mine, operations retreating upward with the water level. The greatest depth that had been reached was 1400 ft. below the plain.

From 1857 to 1903, mining was carried on in a desultory way and, with one exception (when an unsuccessful attempt was made to unwater the mine), was confined to robbing pillars above the permanent water level and resorting old dumps on the Proaño hill.

That a vast amount of mining has been done is shown by the large tonnages of old tailings from the patio process and of the dumps left. Of the former 2,000,000 tons was reclaimed for re-treatment by lixiviation; the dumps contained 300,000 tons of low-grade ore.

In 1900, the Fresnillo company acquired a large part of the tailings and erected a lixiviation plant for their treatment. In 1910, the company extended its holdings to the mine. A 500-ton cyanide plant was partly constructed in 1911–1912 to treat low-grade ore in the Proaño hill. Power for the mill was generated in a gas-engine plant. The company was forced to suspend operations in 1913 by the activities of revolutionary factions and bandits in the State of Zacatecas. This shutdown lasted until January, 1919. In November, 1919, the Mexican Corporation, S. A., obtained a long-term lease on the property, in return for which it agreed to construct a cyanide plant having a capacity of 2200 tons daily, with a power plant of adequate size, and to provide other necessary equipment.

Geology

At the foot of the Proaño hill, there is a large and handsome building, which was the home of the National School of Mines of Mexico from 1854 until 1859. In 1857, Pascual Arenas, professor of mining in this school, wrote a monograph on the geology of the Fresnillo district and the Proaño mine.

The regional formation is mainly sedimentary. A thinly laminated carbonaceous shale is conformably overlain by an irregularly bedded graywacke, the combined thickness of these two members amounting to more than 5000 ft. Because of an almost complete lack of fossil remains, a definite correlation of these sediments presents considerable difficulty. They have, however, been tentatively classified as Devonian.

The shale-graywacke formation was subjected to a gentle folding and later intruded in a scattered and irregular manner by small igneous masses of a medium-basic type. These isolated intrusions appear to be separate offshoots of a deep-seated central mass. A series of Tertiary flows, varying in composition from a trachyte to a basalt, later covered the country. Subsequent erosion practically removed these flows, only isolated fragments remaining. It likewise stripped enough of the shale-graywacke formation to expose many of the isolated intrusions and so wore these away, with one or two exceptions, that they are no longer conspicuous.

The final stage was the deposition of a recent earthy limestone locally known as "caliche." This formation is unconformable with the underlying Devonian. The beds are practically horizontal and are 50 to 75 ft. thick, sufficient to conceal the older formations except where they rise above the plains as hills.

The mineralization of the district was effected in the Tertiary period, following the intrusions and preceding the appearance of the flows, and resulted probably from processes of magmatic differentiation within the mass of the deep-seated central intrusion.

Proaño hill is an outcrop of graywacke. Its elevation above the surrounding area was probably due to upward pressure exerted by the igneous intrusive. During this process the Devonian sediments were severely fractured. Four general fracture systems resulted, within which are comprised all the structures of importance. In addition to the four main groups of fractures, and entirely subsidiary to them, there are found innumerable minor structures, many with strikes roughly paralleling the principal systems, others striking independently in every direction.

The area within which productive veins have been found is 1 mile long by ½ mile wide. The important units of each fracture system meet in the central portion of the hill where the rock was also intensely fractured by subsidiary movements, resulting in a shattered zone 1200 by 400 ft.

The mineral solutions, rising along the main fracture planes, also entered the minor fractures and cracks in all their deviations. They not only precipitated their metallic contents along the fractures but because of the mineralogical composition of the graywacke, effected replacement within the rock. The mineralization was more intense in the immediate vicinity of the principal channels through which these solutions rose. It was along the lines of this main fracture, therefore, that the ore was formed that attracted the attention of former operators.

The valuable metals are silver and gold, the ratio by weight of the former to the latter being 1000 to 1.5. Argentite is the predominant silver mineral and is associated with pyrite and manganese mineral.

The sulfides are primary throughout. A slight secondary action is represented by small quantities of native silver and silver halides, principally cerargyrite, which are found in the upper part of the mine.

No zone of secondary sulfide enrichment is indicated at greater depth. There are, however, evidences that a zoning of primary sulfides will be found, and that argentite will gradually be replaced by primary lead and zinc sulfides.

Oxidation has penetrated with great thoroughness to the lowest horizon so far exposed—420 ft. below the present ground-water level.

The shattered mass within the central portion of the hill and near the surface was sufficiently mineralized by the processes just described to produce a large stockwork deposit, which, on account of conditions favorable to low mining costs, can today be profitably exploited.

The subsidiary fractures and cracks become less numerous in depth: likewise, the amount of valuable mineral deposited outside of the principal fracture systems. The transition from ore to unprofitable material varies as to depth in different parts of the deposit; on the average the bottom limit is 300 ft. below the summit of the hill. The principal vein systems, however, continue productive to the deepest level exposed to date. In the two most important veins, the mineralization was effected in their upper portions—not along a single plane of weakness but over a fairly wide fracture zone. The lateral limits of these zones are fixed by the degree of intensity of the subsidiary fracturing, there being no well-marked walls. The ore limits can be determined by sampling only.

At 350 ft. below the local datum plane, which is 320 ft. below the summit of the hill, one of these veins has an average payable width of 45 ft., and the other a width of 36 ft. As further depth is reached, these mineralized fracture zones appear to be consolidating into formal veins of less width. The other veins now exposed are far more regular in character. They vary in width from 3 to 10 ft., and exhibit a comparative uniformity from the surface to the lowest point at which they have yet been observed, 525 ft. below the datum plane. The small proportion of the quartz in the ore is noteworthy.

So far as known, the pre-mineral faults are few and of little importance. The post-mineral movements are numerous and widespread but, with one exception, they have not displaced the veins enough to interfere either with mining or development operations.

Early Mining Methods

The ore extracted by the Fresnillo company and by the Mexican Corporation, S. A., until the end of 1920 was broken in quarries. The details of the method used were described by D. B. McAllister.[1] A résumé is given to convey a general idea of the method used. No

[1] *Eng. & Min. Jnl.*, May 7, 1921.

mechanical power was available for mining operations; consequently more labor was required than would otherwise have been necessary.

The quarry faces were carried by benches 20 ft. high. Holes were spaced 6 to 10 ft. apart and along a line 6 to 10 ft. back. Double-hand

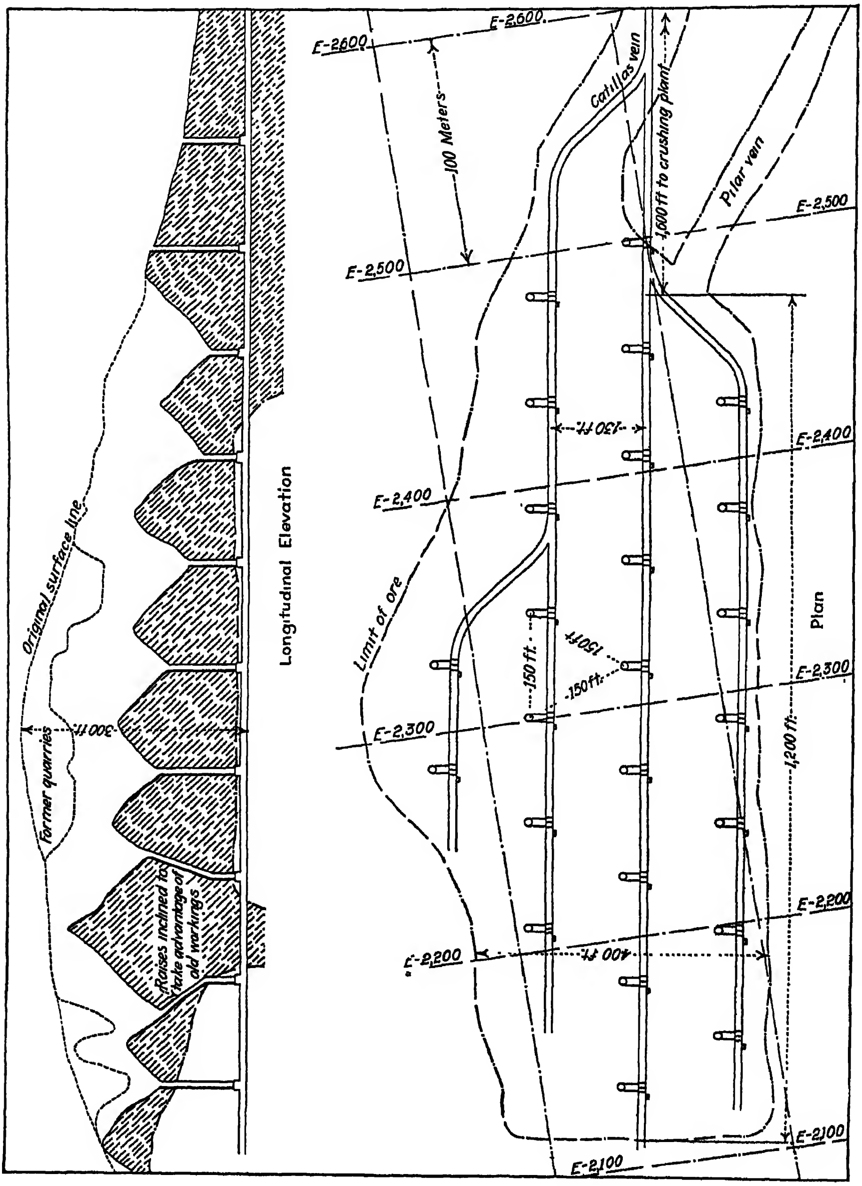

FIG. 1.—ARRANGEMENT OF HAULAGEWAYS AND GLORY HOLES AT FRESNILLO.

drilling was employed for the first 5 ft. of a 20-ft. hole and hand-churn drilling for the remainder. A pair of good miners working on contract would drill a hole in a shift.

The hole was chambered at the bottom, by successive springing, to hold the requisite quantity of explosive. After being allowed to cool, the chamber and about two-thirds of the hole were loaded with ⅞-in., 40 per cent. nitroglycerin, tamped thoroughly to eliminate air spaces. The upper part of the hole was filled with stemming made of sand and clay. A No. 8-X blasting cap was placed about midway in the charge of dynamite and an electric cap near the top. From four to ten holes were fired simultaneously by a blasting machine, all circuits being connected in series. Nitroglycerin dynamite, having a higher rate of detonation than the gelatin or ammonia dynamites, proved most economical as less secondary blasting was required.

The ore had to pass through an 8-in. gyratory; pieces that were too large were broken to the proper size on the quarry floors by block-holing or by sledge hammers. The broken ore was shoveled into a side-dump car having a capacity of 26 cu. ft. and trammed on a 20-in. gage track, by two men, to the mill bins. The main track, which was connected with each quarry, was halfway between the summit of the hill and the glory-hole haulageway. The average mining costs for 1920 were:

Quarry Costs

	Per Ton	
Drilling	$0.17	
Timbering	0.03	
Explosives	0.023	
Tracking	0.03	
Lighting	0.005	
Supervision and miscellaneous	0.04	
Total		$0.298
Quarry to mill bins		0.108
Surface		0.067
Total mining, excluding development		$0.473

Of the above $0.34 was for labor.

Average Duty per Man

	All Operations		Ore Breaking	
Total Tonnage, Month	Total Number Shifts	Tons per Man-shift	Total Number Shifts	Tons per Man-shift
15,800	6176	2.39	2870	5.50

Average Consumption of Explosives per Ton Ore

40 per cent. nitroglycerin dynamite	0.122 lb.
Blasting caps	0.091 caps.
Fuse	0.333 ft.

Glory-hole mining has increased the dynamite consumption per ton of ore broken one-third over the average for quarry work, because

straighter faces and higher benches can be carried in a quarry. The glory-hole system has, however, greatly reduced labor costs.

The construction program of the Mexican Corporation was predicated on the results of sampling the large, surficial ore deposit. At the time the property was acquired only the ore above the level of the plain was accessible for measurement and sampling. It was estimated that the orebody contained 5,000,000 tons of ore, averaging 5.25 oz. silver and $0.20 gold per ton. This would form the basis of future operations, with good chances of developing later substantial tonnages of ore in the underground mine. The mill was built for a daily capacity of 2200 tons, but it has been enlarged to treat 3500 tons a day.

Adoption of Glory-hole Mining

Open-cut mining with steam shovels was considered before selecting the glory-hole method. The factors analyzed and the conclusions reached were as follows:

1. According to comparative estimates made, the cost of mining with steam shovels would be lower than by the glory-hole method. The indicated advantage in favor of the former was questioned, however, as the estimated margin was too small to be within the limits of accuracy of the data taken as bases for the calculations. This advantage was discounted by the greater uncertainty attached to the steam-shovel estimate, because of the absence of steam-shovel data applicable to this orebody.

2. The silver minerals were deposited throughout the ore deposit in a very irregular manner. Sampling had shown that portions of unprofitable material would be found. Whereas large blocks could be handled equally well by both methods, steam shoveling seemed to offer a better opportunity for the selective mining of small bunches than the glory-hole method.

3. The glory-hole method is an extremely simple operation and does not require skilled labor, whereas experienced foreign shovel men and cranemen would be needed in steam shoveling, and a sure supply thereof could not be counted on.

4. A reserve of 10,000 tons of broken ore could be carried in the glory holes to insure continuous milling operations at full capacity. On account of the low cost, the interest charge on the money represented in breaking this ore would be inconsequential.

On the other hand, reductions in the milling rate might follow serious accidents to steam shovels or to the haulage equipment on steep grades and curves required. The mill has treated 1,600,000 tons of ore, and not once since it was started has any reduction in tonnage been necessary for lack of ore. The advantages of the glory-hole method over

steam-shovel mining were considered enough to outweigh the possible disadvantages.

The haulageway system for the extraction of the ore was placed low enough to serve most of the ore deposit and at the same time leave enough

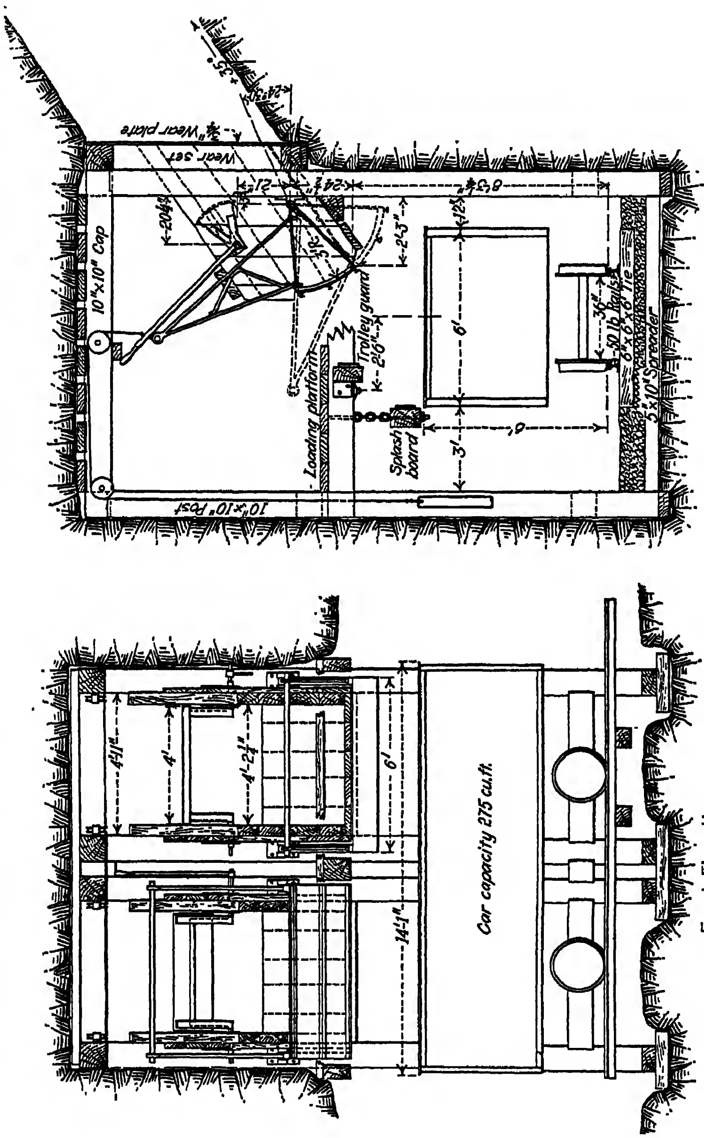

Fig. 2.—Loading Chute used in Glory-hole Practice at Fresnillo.

of the hill slope below the portal for a suitable mill site. The adit is 2200 ft. long and is parallel to the longitudinal axis of the orebody. The distance from the portal to where it enters under the orebody is 450 ft.; there it branches into parallel haulageways 130 ft. apart. The radius of

each turnout is 100 ft. The standard haulageway section is 10 ft. wide by $9\frac{1}{2}$ ft. high. Except at the loading chutes, little timber is used. Fig. 1 shows the arrangement of the haulageways with respect to the orebody.

Loading Stations

Along each haulageway, loading stations are spaced at intervals of 150 ft. To provide room for the timbering, the back of the drift, for a length of 14 ft. is raised to a height of 17 ft. above the rail. A pilot raise, around which the loading pocket is subsequently stoped, is then driven high enough so that the station and chute timber can be placed without danger of being subsequently blasted.

Double chutes are built at each station, so that a car can be filled with one spotting, and should one chute become blocked by large boulders, loading can be done from the other. Underswung, hand-operated arc gates are used in conjunction with radial check gates. Most of the control is secured by the latter, the underswung gate being used to start and stop the flow of rock. Without the check gate, the pressure on the underswung gate is enormous and manual operation thereof is almost impossible. The operating platform is above the trolley line, which makes for safety and ease in loading the trains.

Fig. 2 shows details of the loading chute, which is similar to one used successfully at the Alaska Gastineau mine. The chute details and the relation of the underswung gate to the chute bottom have been modified somewhat since operations started. The gate works best when the chute bottom has a slope of 35°, and the arc is described as shown on the drawing. The gate is overbalanced by a counterweight, so that the flow of ore is easily and quickly stopped; this requires a greater effort in opening, but permits a quick and positive cut-off.

The loading arrangements have proved successful, considering the fact that the ore varies in size from a good proportion of fines to a maximum of 30 in. The underswung arc gate handles the large boulders relatively easily and with little spillage of the fine material.

Air cylinders for operating the gates were considered, but were not adopted for the following reasons:

1. Only three extra men per shift were required for the hand-operated gates.

2. Air cylinders would always require a certain amount of upkeep and are wasteful of air, especially when leaks are not promptly stopped.

3. During a temporary air shortage arising from compressor troubles or a break in the pipe line, loading operations would be at a standstill, if manipulation of the gates was dependent on air cylinders.

After the loading station has been timbered and the loading chutes are completed, the pilot raise for the loading pocket is extended to a

height of 35 ft., measured above the floor of the haulageway, and is then stoped out to provide storage for 100 tons of broken ore.

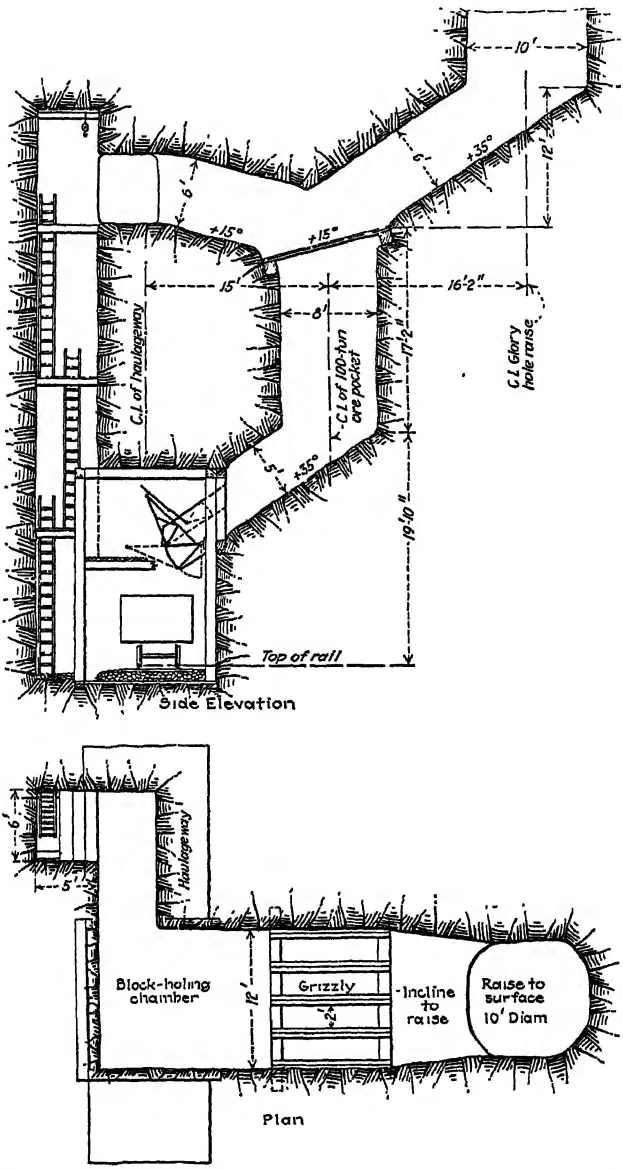

FIG. 3.—GRIZZLY, BLOCK-HOLING CHAMBER, AND ORE POCKET.

The grizzly is set directly over the loading pocket on an inclination of 15° with the horizontal. The grizzly bars, each formed of two 100-lb. rails riveted to a 10 by ⅝-in. plate, are spaced 24 in. apart. Each bar

can be removed separately and with ease; credit for the design is given the New Jersey Zinc Co.

From the grizzly, the main raise to the surface is driven at an angle of 35° for a distance of 18 ft. before it is turned vertically. This is a desirable feature, as the ore falls on a rock-bottom incline and slides or rolls over the grizzly. The excavation above the grizzly chamber and the incline are so designed that the broken ore cannot flood the entire grizzly. The big boulders have a tendency to roll to the lower side, where they can be easily broken by block holing. The success of the

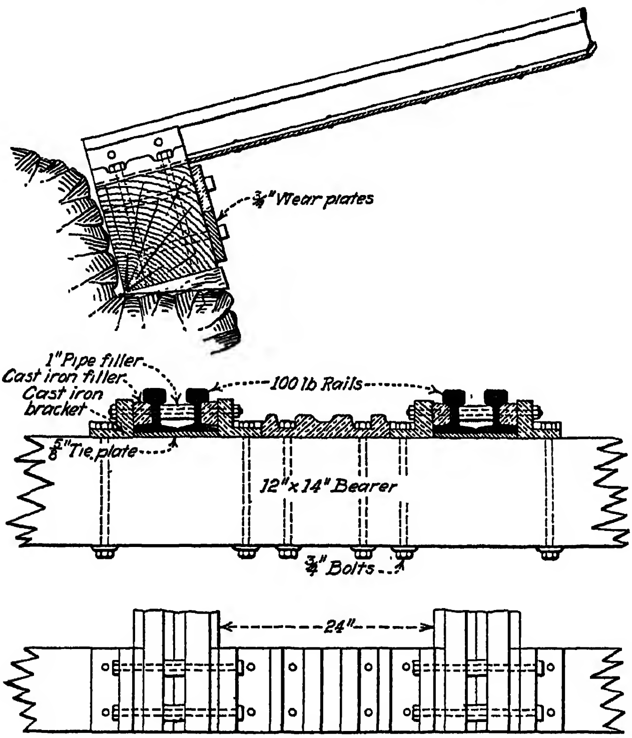

Fig. 4.—Detail of grizzly shown in Fig. 3.

design has been fully demonstrated by the ease with which the ore pockets are kept full and the lack of wear on the grizzly bars. As much as 2000 tons of ore has been drawn from one loading station in 24 hr. To date only two bars out of ninety-five in service have been replaced. The maximum tonnage that has been handled so far by any one grizzly is 235,000 tons.

No timber is needed to support the ground in the loading pockets or raises. The vertical raise to the surface has a circular section, the diameter being 10 ft. Details of the ore pocket and grizzly arrangement are shown in Fig. 3 and of the grizzly in Fig. 4.

On the opposite side of the haulageway, and 6 ft. from the loading station, a raise is driven from the track level to a point 10 ft. above the floor of a short sublevel connecting the raise with the block-holing chamber. Head room is thereby furnished for raising and lowering grizzly bars. A snatch block is hung at the top of the raise to facilitate the work. A good ladderway with landings at each ladder provides easy access to the block-holing chamber. A clear space is left in each manway for hoisting the grizzly bars. A 1-in. pipeline furnishes air for drilling the large boulders.

The manway is offset from the block-holing chamber by the short connecting drift, so that it is out of the way of flying rock when blasting. The floor of the block-holing chamber slopes down to the grizzly to stop large boulders rolling down the incline at or near the foot of the grizzly.

TABLE 1.—*Average Cost per Foot Advance*

	Feet	Drilling	Timbering	Explosives	Loading and Tramming	Total Cost: Labor and Bosses	Supplies	Power	Total
Haulageways........	5,026	$ 9.19	$ 2.58	$ 3.11	$ 3.74	$ 10.65	$ 6.18	$ 1.79	$ 18.62
Glory-hole raises.....	3,194	7.92	0.84	1.79	0.86	8.16	2.92	0.33	11.41
Ore-pocket inclines...	1,337	10.34	3.89[a]	1.74	0.91	11.52	14.78	0.58	26.88
Sub-levels and manways..............	1.074	4.99	2.18	0.54	0.69	5.91	2.19	0.30	8.40
Miscellaneous development...........	1,109	3.48	0.64	1.04	0.61	3.91	1.63	0.23	5.77
Tracking (5,026 ft.)[b]..						1.66	1.76		3.42
Lighting carbide......							0.38		0.38
Totals and averages	11,740	$ 8.04	$ 3 18	$ 2.16	$ 2.05	$ 9.70	$ 6.62	$ 0.98	$ 17.30
Glory-hole loading station..........	19 stations	$29.22	$917.88[c]	$20.38	$18.34	$336.99	$634.56	$14.27	$985.82

[a] Includes grizzlies and temporary timbering. [b] Rails and splicebars charged to equipment. [c] Includes iron work.

Tons ore recoverable by drawing only, 6,000,000; total cost of preparatory work, $221,784.46; cost per ton $0 0370.

DRILLING PRACTICE

The benches are 9 ft. high and the floors are kept as close to the horizontal as possible. Holes are spaced 3½ to 4 ft. apart, and are drilled vertically from 3 ft. to 5 ft. from the face of the bench. This system of using comparatively shallow holes spaced fairly close has proved more economical than deep holes with heavy loads. The reasons are:

1. The ore is broken to better size, thereby reducing the expense of secondary blasting; also glory-hole raises are less liable to be blocked with large boulders.

2. Holes can be drilled dry to a depth of 10 ft., whereas water machines and water lines would be required for deep holes. The water lines would not only be an added expense, but would prove a nuisance in the glory holes.

3. Labor for loading and blasting shallow holes is less per ton of ore broken.

4. Local machine men are more efficient in drilling short holes.

5. Less drill steel is broken and stuck with short holes; also, pieces are lighter and, therefore, more easily distributed to and collected from the working faces.

Deeper holes and wider spacing gave more tons per foot of hole drilled and a slightly lower powder consumption per ton broken, but the final results showed a saving in favor of the system used.

A slope angle of 50 to 60 deg. for the glory-hole sides gives the best results. For safety reasons an endeavor is made to keep holes slightly filled above the collar of the raise. This in a measure increases the cost of secondary blasting, as there is considerable breaking effect obtained in the falling of the ore after blasting, varying with the distance of fall.

Holes are drilled on the day shift of eight hours. One or two crews, depending upon the production desired, are worked in a glory hole. Each crew consists of a drill runner, his helper, and two barmen, one acting also as a "nipper." All work in the glory holes, except for the blasting and pipe work, is contracted. The contractor, who is the drill runner, furnishes his own crew, and is paid at a rate per foot drilled, this price covering the work of "barring down" and "nipping." The average footage drilled per shift is 155 ft. It was considered impracticable to base the contract price on the unit of tons of ore broken, largely because part of the rock broken in one glory hole often falls into one or two adjacent glory holes.

The contract price in effect was determined after operating on the day's-pay system for nearly a year. During that time the drill runners, who were entirely unfamiliar with machine drills at the start of operations, were given special supervision before contracting the work. The need for this precaution is obvious, as it was desirable to fix a price for once and all that would prove equitable to both the company and the men, otherwise confidence in the contract system would be destroyed by subsequent changes.

The weakness of the present contract system lies in the fact that the contractor in most cases pays little, if any, more to his crew than formerly and pockets the profits himself. It is preferable to distribute the earnings among all the workmen, not equally, but in proportions based on the relative wage values assigned to the different classes of work. On the other hand, under the present system the contractor is able to reward his men according to individual service rendered, whereas any profit-

sharing plan devised by the company would have to be based upon a fixed percentage for each occupation, irrespective of the relative work done. Theoretically the existing system is the better, but it is believed that a satisfactory profit-sharing system would make for more general contentment among the men. Careful study is being given to devise a plan that will prove to be practicable and fair to all concerned.

Hollow hexagonal ⅞-in. steel is used entirely. The breakage thereof is undoubtedly higher than it would be for 1-in. steel of the same section, but the ⅞-in. size is lighter, and, therefore, easier to distribute. No tests have been made to determine drilling speeds of the two sizes, but it is thought that the smaller steel would drill faster.

Table 2 shows the steel consumption from Aug. 22, 1921, to April 15, 1922:

Table 2.—*Drill-steel Consumption*

Tons ore broken during the period	455,020
Pounds drill steel consumed	9,390
Pounds drill steel consumed per ton ore	0.021
Cost of drill steel per ton of ore broken	$0.0028
Feet of hole drilled	344,900
Pounds drill steel consumed per foot of hole drilled	0.027
Cost of drill steel per foot drilled	$0.00364

Steel consumption is due mainly to stuck drills and breakage. Fractures filled with soft vein material are mainly responsible for the former. An effort is made to recover stuck drills by supplying a crew with special wrenches which fit around the steel directly underneath the collar and are struck upward by a hammer. An average of 3200 ft. is drilled for every steel that cannot be withdrawn. This does not represent a total loss, as the steels are recovered in the chute. They are straightened and re-used, when possible to do so, or are made into starters or sampling moils. Occasionally a drill steel or bar, through carelessness, will be dropped into the glory holes.

Records show that 330 ft. of hole is drilled per broken steel and that 64 per cent. of the breaks is in the bits, 26 per cent. in the shanks, and 10 per cent. in the center.

Owing to the ease with which the ground is drilled, and to the fact that part of the steel loss would be the same regardless of the grade of steel used, the determining factor in selecting a brand is price more than quality. For this reason cheap Swedish steel has proved more economical than other brands tested.

The double-taper cross bit, 14° and 5°, is standard. Although the ground is suited to $\frac{1}{16}$-in. changes in gage, a ⅛-in. variation was adopted, as less skill in sharpening is necessary. Although the drill sharpeners can be trained to make well-formed bits, it is difficult to hold them to any fine degree of accuracy.

A 24-in. change in length is used, making five pieces of steel to a set. The gage of the starter is $2\frac{1}{8}$ in., and of the finisher $1\frac{5}{8}$ in.

Drilling in the glory holes is performed by Ingersoll-Rand DCR-13 drills of the Jackhamer type. The weight of this model is 55 lb. Time studies show that the machines are actually drilling 60 per cent. of the shift. The average duty per machine-drill shift is 150 tons.

For block holing, Ingersoll-Rand BAR-33 Jackhamers were selected on account of their light weight. This model weighs only 21 lb., and is, therefore, conveniently carried in the glory holes and underground, from one block-holing chamber to another.

Repair costs per machine-drill shift are shown in Table 3.

TABLE 3.—*Repair Costs per Machine-drill Shift*

Model	No. in Service and in Repair Shop	Overhead	Labor	Supplies	Power	Total
DCR-13	30	$0.002	$0.039	$0.120	$0.001	$0.162
BAR-33.......	6	0.0005	0.009	0.003	0.0005	0.013

The sharpened steel is distributed to the "nippers" from a small shed close to the glory-hole operations, and the dull steel is returned to it. Both the sharp and dull steel are hauled on muleback to and from the drill-sharpening shop, which is near the portal of the adit, the hauling being contracted. This shop also serves the deep mine. From the time the drill steel is deposited in a large sorting table until it is stacked in the finishing racks it is routed through the shop without any unnecessary handling. The shop arrangement and the sharpening and tempering methods follow closely the practice in the best shops of the large mines in the Southwest. A detailed description therefore is unnecessary.

Sharpening is done by two Waugh Model No. 8 Sharpeners. Drills are heated for sharpening and for tempering in separate furnaces, each built locally according to an original design of Tom Davies, of the Ingersoll-Rand Co. The furnaces run quietly and uniformly, emit little smoke, and have given better results with crude oil than any other type tried. A small blower supplies air at 3 lb. pressure to all the forges and furnaces within the shop.

All sharpening is done on the day shift. The shop personnel consists of one Mexican shop foreman, two sharpeners and two helpers, and one temperer and helper.

STEEL SHARPENING DATA

	Labor	Supplies	Power	Total
Cost per piece sharpened..	$0.0162	$0.0136	$0.0128	$0.0426
Cost per ton ore broken..............				0.0057
Pieces sharpened per sharpener-shift..				300
Crude-oil consumption, gallons per burner hour.				2.4

Blasting Practices Developed

The blasting crew comes on at noon, and by 5 p.m. an average number of 400 holes are loaded and ready to blast. Some blasting, principally block-holing of large boulders, is done at noon. The crew consists of a head powder man and six helpers. The men work on company account, but eventually this will be contracted on a per-ton basis to include explosives.

Single-tape safety time fuse and No. 8-X caps are used for detonation of explosives, except in those holes around the rims of the glory holes and first benches where electric blasting caps with 12-ft. wires are fired. A summary of results obtained with different grades of DuPont explosives under actual operating conditions is given in Tabl 4.

Table 4.—*Summary of Blasting Experiments with Different Explosives*

	Durox	Dumorite	40 Per Cent. Nitroglycerin	Monobel	Quarry Special	40 Per Cent. Ammonia
Tons ore broken during test......	80,048	85,312	20,988	17,312	16,516	4,100
Tons ore per pound of dynamite..	6.3	6.1	5.1	4.7	4.6	4.00

Durox is rated according to strength as 35 per cent. nitroglycerin, and its velocity or rate of detonation is about the same. Durox is cheaper than straight dynamite, for, being less dense, the rise in the hole for equal weights is greater, and thereby distributes the explosive action over a greater length of the hole. Also, the blasters are inclined to use less, by weight, of the bulky powder. The lower grade quarry explosives, such as Quarry Special, failed to break the ground sufficiently. Black powder was tried, but, owing to its slow velocity, with consequent heaving instead of shattering action, was found unsuitable for glory-hole mining.

An average of 1½ lb. of explosive, in cartridges 1⅛ by 8 in., is loaded in a 9-ft. hole and tamped with clay stemming, transported, for convenience and to preserve the moisture, in paper cartridges made from the paraffined packing paper of the dynamite cases. The weight of ore broken per cap is 9.3 tons; 1.2 ft. of fuse is used per ton of ore.

All secondary blasting is done by block holing. A hole 6 to 12 in. deep is drilled by machine in any rock that is too large to pass through the grizzly openings. It is loaded with one-third to one-half of a 1⅛ by 8-in. stick of Durox dynamite, and detonated by a No. 6-X cap. Bulldozing is employed only in unblocking raises. By breaking the ore in the glory holes to a comparatively small size, explosive consumption in secondary blasting as light as shown in Table 5.

TABLE 5.—*Explosives Used for Secondary Blasting*

	Tons Ore Broken per Unit Used	Cost per Ton
$1\frac{1}{8}$ by 8-in. Durox dynamite, pounds	132 0	\$0.0015
No. 6-X blasting caps, each	10.1	0.0006
Single-tape fuse, feet	5 3	0.0006
Total		\$0 0027

TRANSPORTATION

The track, 36 in. gage, is laid with 50-lb. rails, on 6 by 8-in. by 6-ft. ties, spaced 24 in., rock ballasted, with a grade of 0.4 per cent. in favor of the load. The roadbed was built with the utmost care, and is maintained in good condition, so that delays in transportation on account of derailments are rare. Switches are of the split or movable-point type, and are standard for a No. $3\frac{1}{2}$ frog, 36-in. gage. The points are operated by a standard railroad switch stand of the upright type.

The haulage system is electrified. No. 0000, hard-drawn, grooved, copper wire carries 250-volt direct current 8 ft. above the rail. The trolley-wire suspensions are 30 ft. apart and supported by 6-in. expansion bolts or by span wires, depending on the height of the roof above the track. At the loading chutes, ordinary screw-caps are used. The rails are bonded with compressed-terminal bonds concealed by the splice bars. To protect the locomotive armatures against high-voltage disturbance, each locomotive is equipped with an oxide lightning arrester. The motor-generator is protected by a lightning arrester connected to the feeder cable.

A train of nine cars is hauled by an 8-ton trolley-type Baldwin-Westinghouse locomotive of bar-steel frame construction. The latter is equipped with steel tires and has a rated drawbar pull of 2000 lb. at a speed of 6 miles per hour. The maximum gross trailing load is 165 tons. Two of these locomotives have hauled 1,600,000 tons to date, and have traveled a total distance of 13,600 miles with only a nominal amount of repairs. The original steel tires will be removed shortly to be re-turned for the first time.

The car for tramming ore has a rigid flat-bottom body, the capacity of which is 275 cu. ft. The truck has four wheels; the wheel base is $6\frac{1}{2}$ ft. The cars are equipped with three-quarter M.C.B. swivel couplings, and are unloaded in a one-car, rotary car dumper without uncoupling from the train. The drawbars work against 1-in. springs; these act in tension or compression to decrease the tractive effort when the train is

started or to absorb the jolts when stopped. Single 4 by 7-in. Hyatt roller bearings are inclosed in cast-steel, dust-proof journal boxes. Each journal box is supported on two 7/8-in. springs within a malleable-iron pedestal. The bearings are filled with Slo-Flo lubricant once every two months. The cost of lubrication per ton of ore hauled is a negligible quantity. The wheels, 20 in. diameter, are made of chilled cast iron and pressed on 4½-in. open-hearth steel axles. The average life of a wheel is eleven months, during which time 9500 tons of ore is hauled per wheel. The body plates opposite the chute doors are lined with 20-lb. rails placed vertically 6 in. apart. The bottom plates are overlaid with 1¾-in. oak boards, which are in turn protected by ¼-in. steel wearing plates.

The cars are satisfactory in every respect. They are subjected to extremely hard usage, because of the size of material handled and the large tonnage (about 200 tons) trammed per car per day.

The rotary car dumper originally installed was too light for the service. A dumper of similar design, but of much heavier construction, was fabricated locally and is giving entire satisfaction. It is operated by compressed air. The air cylinders are connected to the compressed-air main, and also to an independent 100-cu. ft. compressor, which can be used to operate the car dumper in case of an air shortage.

Two trains of nine cars each handle the production. A motorman and two brakemen comprise a train crew. The brakemen and one chute tender, who assists both crews, load the cars. Two men operate the underswung arc gate, and the third works the check gate. A policeman's whistle is used to signal the motorman. In addition to the train crews, the following men are employed on each shift: one car-dumper operator, one switchman, three men to repair chutes and to timber, one man to clean tracks, and a shift boss, who supervises the block holing on the grizzlies as well as the loading of the trains.

The empty incoming train and the outgoing loaded train pass each other on a double-tracked portion near the crushing plant. The unloading time is governed by the crushing rate, as the storage capacity of the course-ore hopper above the 30-in. gyratory crusher is practically nil. An average time analysis of the different tramming operations involved shows the following for an average tramming distance of 200 ft:

Time Study of Tramming Operations

	Minutes
Loading	11
Transporting to switch	6
Waiting at switch	3
Switch to crushing plant, dumping and return to switch	25
Returning empties to loading chutes	5
Total for round trip	50

No ore tramming is done from 7 to 11 a.m., for during this period the crushing plant is thoroughly inspected and all repairs necessary are made; also the waste is hauled to the dump. Experience has proved that the crushing plant can be kept in better condition and less time is lost when inspections and repairs are made regularly each morning. Detailed tramming costs are shown in Table 6.

TABLE 6.—*Tramming Costs per Dry Ton Hauled*

Operation	$0.0174	Loading chutes	$0 0033
Cars	0.0071	Labor	0 0218
Track	0.0020	Supplies	0.0073
Locomotives and motor-generator	0.0012	Power	0.0049
Car dumper	0.0013	Total	$0.0340
Trolley lines	0.0017		

When it becomes necessary to mine unprofitable material in any glory hole, the broken ore is drawn out before the waste is blasted. Hand-operated side-dump cars of the contractor's type are used to haul the waste from the loading chutes to the waste dump. The capacity of each car is 4 cu. yd.; the over-all height is the same as that of the ore cars.

Only a small part of the mine superintendent's time is required to supervise the glory-hole operations, the remainder being devoted to the more complex underground work. The supervision is performed mainly by a foreman, who directs the daily routine work of drilling and blasting. Shift bosses, under his direction, supervise the loading and tramming. The total charge for supervision is small.

SAMPLING

On account of the low silver content, and the equally oxidized appearance of the ore and unprofitable material, it requires experience and close observation to distinguish between them. At the beginning of operations, a large crew of samplers was employed to take cuts along each bench; later only doubtful faces were sampled; now the foreman takes a few samples himself whenever in doubt as to the ground.

Because there is no hold-up by storage ahead nor within the crushing plant, it is possible to take automatic samples of car- or train-load lots in the sampling plant after crushing. A large bulletin board, placed at the car dumper, is visible from the sampling plant. When a train load reaches the crushing plant, large placards are posted to show the numbers of the glory holes from which the ore was drawn and the number of cars. A bell in the sampling plant notifies the sampler of the arrival of the train. Independent automatic samples are taken of each glory-hole lot, in addition to the general head sample. The average grade is determined by taking the tonnages drawn from each glory hole and the respective grade

thereof checks the head sample to 0.5 per cent. The tonnage figures are obtained by weightometer, which is checked weekly by weighing several train loads on thet rack scale. The system gives an accurate assay value of the ore drawn daily from each glory hole, eliminating the need for chute samples, which are unreliable.

When the grade of ore from a glory hole drops below the economical limit, the material is handled as waste. Thereafter, every fifth carload is drawn into an ore car and delivered to the crushing plant; an automatic sample is taken of this lot to check previous sampling. When a large tonnage of unusually low-grade material must be removed, the sampling interval is increased. A close check on the routing of waste and ore is made possible by this system. As the grade of the unprofitable material usually is close to the commercial grade, the treatment of every fifth carload of waste represents an inconsequential loss. The expense of taking hand samples would more than offset the loss involved in treating whatever unprofitable material is milled, and the results obtained by this method would be less reliable.

Amount of Air Consumed

An exhaustive series of tests made to determine the actual consumption of air shows the following:

Conditions	Cu. Ft. of Free Air per Min. at 80 Lb. Pressure: Unit	Total
Average consumption 1 DCR-13 drill when working..........	96	
Average consumption 1 DCR-13 drill during 8-hr. shift.......	61(*a*)	
Average consumption 30 DCR-13 drill during 8-hr. shift......		1830
Average consumption 6 BAR-33 drill during 8-hr. shift.......		30
Average consumption rotary car dumper during 8-hr. shift....		40
Average consumption 1 Waugh Model No. 8 drill sharpener during 8-hr. shift....................................		55
Total..		1955

(*a*) Includes air used in blowing out holes.

The above covers any leakage at the machines but not in the air lines.

It has been proved that an air pressure of 80 lb. at the drills is the most economical pressure to carry, as the ground drills easily. By increasing the speed of drilling, higher air pressures decreased the cost of drilling, but not enough to compensate for the heavier charges for power, machine-drill repairs, and steel breakage resulting therefrom. The sizes of the main and branch air lines are carefully figured so that the drop in pressure between any drill and the compressor will not exceed five pounds.

Labor Statistics

The distribution of labor and the tons broken for each group averages as follows:

	Number of Shifts per Day	Per Cent. Total	Tons per Man-shift
Development (variable)	45	16 1	79.3
Drilling—contract	103	36 5	34.7
Drilling—day's pay	18	.6.4	197.2
Primary blasting	7	2.5	510.0
Secondary blasting	48	17.1	74.3
Timbering			
Loading and tramming	42	15.0	85.0
Air piping	3	1.1	1190.0
Steel sharpening	4	1.4	893.0
Tracking	4	1.4	893 0
Miscellaneous surface labor	7	2.5	510 0
Totals and averages	280	100.0	12.8
Totals and averages for men charged to ore breaking	179	63.6	20.0

Mining Costs Summarized

Average mining costs per month are given in the following:

		Per Ton Ore
Development (based on 6,000,000 tons developed)		
Drilling		$0.017
Timbering		0.009
Tracking, not including rails		0.003
Loading and tramming		0.004
Explosives		0.004
Lighting		
Total		$0.037
Ore breaking		
Stripping	$0.005	
Drilling	0.072	
Secondary blasting	0.018	
Timbering		
Explosives	0.046	
Lighting	0.002	
		$0.143
Loading and tramming		0.034
Surface expense(*a*)		0.003
Engineering and drafting		0.002
Sampling and assaying		0.003
Supervision		0.004
Total operating expense		$0.226

(*a*) Miscellaneous only. Drill-sharpening, compressed air, air-piping and machine-drill repairs are distributed accounts, which are apportioned against the direct accounts in each case according to the relative benefits received.

The costs given in the foregoing table are segregated as follows:

	Labor	Supplies	Power	Total
Development	$0.022	$0.015	$0.002	$0 037
Ore breaking	0.059	0.057	0.027	0.143
Loading and tramming	0.020	0.007	0.005	0.034
Other accounts	0.009	0.003		0 012
Totals	$0.110	$0.082	$0.034	$0.226
Percentages	48.7	36.3	15.0	100.0

Of interest in connection with glory-hole mining is the low cost of equipment for producing 3500 tons per day.

	Labor	Supplies	Machinery	Eng. and Supt.	Total
Drill-sharpening shop	$1,272.28	$1,659.94	$ 3,709.62	$ 44.01	$ 6,685.85
Mine compressor (including motor)	612.01	546.62	12,846.56	136.05	14,159.24
Compressor building	993.23	1,312.20	213.06	155.48	2,673.97
Electrical installation	187.33	126.85	1,895.58	33.65	2,243.41
Rotary car-dumper	805.37	329.30	9,886.45	290.30	11,311.42
Permanent track (material only)		1,459.44	13,912.05		15,371.49
Electrification of tracks		697.30	3,863.23	29.76	4,590.29
Motor-generator set	60.37	17.84	2,684.17	25.11	2,787.49
Electric locomotives	40.47	11.51	13,501.33		13,553.31
Ore cars	37.03	195.07	30,980.50		31,212.60
Waste cars			11,871.86		11,871.86
Air lines			13,018.24		13,018.24
Air drills			9,924.72		9,924.72
Locomotive-repair shed	140.08	101.83		31.20	273.11
Totals	$4,148.17	$6,475.90	$128,307.37	$745.56	$139,677.00

Most of the equipment was purchased when the highest prices were prevailing after the War. All renewals are absorbed by operation and covered by the costs given in the foregoing summary of mining costs.

Conclusions

The system of drilling short vertical holes closely spaced is unquestionably the best one for breaking the ore in glory holes where labor is inefficient. The tonnage broken per foot of hole drilled is low, but the final result is better than when deep holes are drilled at greater intervals.

The duty per machine shift could be increased considerably by drilling deep holes with a machine mounted on a tripod; also, less barring and shoveling would be required to secure a clean face for drilling, but more secondary blasting would be needed, and undoubtedly the raises would be choked more often. At Fresnillo, a mounted machine is out of the question, as too much time is lost in starting and drilling a hole. The jackhammer type of drill requires less skill to operate than the mounted machine, and, therefore, is more suited to the local labor.

The ground stands well without support, making it well suited to the glory-hole system, for the reason that the raises need not be timbered, and hence require no subsequent attention.

A glory hole having a diameter of 150 ft. has a capacity of 15,000 tons per month. The production is limited by the rate the ore can be drawn through the grizzlies. The development work was completed at the outset to provide enough glory holes to assure production at all times, even though a number of glory holes were non-productive at one time on account of mining waste, choked raises, or broken grizzlies. As a result there has been nothing to interfere with the loading and tramming, this making for few delays and less confusion.

Glory-hole methods are most successful where there is little rainfall and no snow. The climate at Fresnillo is ideal. The rainy season extends from July into September, but there are only a few heavy rains. When these last for several day sthe capacity of the crushing plant has been reduced as much as 75 per cent., mainly because the vertical Symons disk crushers will not handle wet ore. During wet weather, the glory-hole raises are kept empty, otherwise the broken ore packs and blocks them; also, the fine material becomes like mud. Slippery working faces make glory-hole work dangerous. When this condition exists, drilling is done only on the rims of the holes.

Few accidents have occurred, and only one of these was fatal. The accident rate is much lower than for underground mining. Steep slopes and lack of proper attention in barring down loose ground make for more accidents. The men are required to use ropes, but infractions of this rule are hard to prevent.

Apart from natural conditions, the loading arrangements are largely responsible for the success attained in glory-hole mining. Even these, however, would not prove satisfactory unless the ore were broken to suitable size by the primary blasting.

I am indebted to Mr. C. E. Prior, mine superintendent, for his assistance in preparing this paper, and to Mr. Max Dixon, geologist for the company, for the geological data.

DISCUSSION

Charles A. Mitke, Bisbee, Ariz.—Some years ago, when the examination was made, we were much impressed by the hill known as Proaño Hill, and when making a study of that mountain was the first time I had seen a silver porphyry. It was the case of a product with a large tonnage, low grade, with bands of waste interspersed with the ore; it was not one clean block of solid ore by any means. There were lots of faults and fissures; and while there was a certain uniformity in the hardness of the rock there was great variety in various parts of the mountain.

Part of the mountain was honeycombed by those who sought the high-grade ore during the preceding two centuries. In formulating definite plans, the company had independent engineers study the mountain and give estimates, methods of mining, and the general layout of the development; everything complete with estimates on costs and so on.

Milling was handled in the same way; and it was not long after the plant had started that everything was in working order and the plant was able to surpass the estimated costs of the various engineers. Even with changed conditions they are still able to work with a very low mining cost.

B. B. Gottsberger,* New Haven, Conn.—Does all that ground stand unsupported?

E. R. Thomas.—There is nothing to support. These drifts, stations, holes, and operating drifts stand without support, with the exception of an occasional stull across the back of the tunnel itself.

B. B. Gottsberger.—How is it, then, that the ground, although standing without timber, appears to be quite easy to drill and break?

E. R. Thomas.—It is a soft rock and is pretty well fractured. The ground is dry and is not working ground. That kind of ground in a stope, assuming that the ground was pretty well arched, would stand a pretty good stress without caving, although it is nearly soft.

Enoch Perkins, Wharton, N. J.—Why do you use double instead of single chute gates? Fig. 2 shows two chute openings 4 ft. in the clear; why not use a single chute opening, say, 6 ft. in the clear? The Alaska Gastineau Co. changed the chute opening from 5½ ft. to two single chutes of 4 ft., stating that in case one chute is blocked the other can be used and the car loaded. Would not a chute be less liable to block with a single 6½-ft. gate instead of the two 4-ft. gates?

A. H. Rogers.—Is the main part of the chute vertical, as indicated at the top of the diagram?

E. R. Thomas.—From that point to the surface, the chute is vertical. The Fresnillo hill is honeycombed with old workings; so sometimes, where they could take out the ore by breaking into an old raise, they would do so rather than go up straight, but the design of the work calls for a vertical chute right up to the surface.

Enoch Perkins.—Why vertical instead of on a 65° pitch? Would not a 65° plane be less liable to hang up than a vertical chute raise?

E. R. Thomas.—I do not think so; it would call for four raises instead of one.

* Professor, Mining Engineering, Yale University.

ENOCH PERKINS.—I do not see how.

E. R. THOMAS.—Assuming that the glory hole on the surface has a diameter of 150 ft., you would need four raises to concentrate at the central point; with one vertical raise, you come out into the center of the circle. Possibly, in starting to break at the collar of the raise some care should be taken not to break too many chunks for the first few rounds of ground. After that, the slope of the glory hole is easier to handle, being just deep enough that the big chunks will not lodge on the sides. With ground fractured as it is at Fresnillo, after the blast travels 10, 15, or 20 ft. it will break up fairly fine at the first bump it strikes, so that there is no trouble.

ENOCH PERKINS.—How will you recover the ore in the chute raise pillars?

E. R. THOMAS.—Fresnillo is almost a true Mexican vein system. Underneath this hill there are workings on a vein system. At present, the old mine is practically in water and enriched so that when this ground is stoped down to this point they simply underhand stope it and break it down to the stopes below and draw it out for the shaft.

CHARLES A. MITKE.—The hardness of the ground affects the size of the openings, raises, and so on. The average rock down there is probably not quite so hard as the average steam-shovel rock at Ajo, and New Cornelia, but is probably similar to the Sacramento Hill at Bisbee, and is very much firmer and harder than that of the Ray Consolidated, the Inspiration, or the Miami Copper Co. Therefore, the size of the raise must be taken into consideration when considering the amount of rock that has to be taken through. This tunnel level is about 300 ft. below the top of the mountain. If, in certain localities, the rock is not so firm, the raises must be put through as small as possible because after several hundred thousand tons are passed through, these raises will grow to be 10, 15, or 20 ft. in diameter. That, I think, to a large extent accounts for variations in sizes of chutes.

B. B. GOTTSBERGER.—The average cost per foot of advance of all the underground development work here is given as $17.30. Considering the cheapness of all of these other operations, particularly their final result, also the fact that apparently all of this underground work stands up without timber, is not that a rather high development cost per foot?

CHARLES A. MITKE.—That includes the cost of the tunnel, which is about 10 by 10 ft., and is partly timbered; there was not very much raising.

ENOCH PERKINS.—There is one comparatively large item—that of $26.88 per ft. ore-pocket inclines. That is, no doubt, yardage reduced to feet, which would help bring up the average.

E. R. Thomas.—When that development work was done, or the greater part of it, the only power plant consisted of six gas engines running on charcoal. While the mill was under construction, this new development work was pushed ahead, regardless of cost, which probably had more to do with the cost of drifting than anything else. If that drifting had been done with the present power plant, the work could have been done for much less money, particularly with the present organization. At that time, however, work was done by hand, which also added to the apparent high working cost. If you were to contract that same work today, the company supplying the air, steel, and so forth, the cost of labor and powder on a drift of that kind should not exceed from 30 to 35 pesos, which means about \$15 to \$17.50 per meter.

Mining Methods at Mascot Mines, Tennessee

By H. A. Coy* and James A. Noble†, Mascot, Tenn.

(Birmingham Meeting, October, 1924)

The Mascot mines of the American Zinc Co. of Tennessee are situated at Mascot, Tenn., 14 miles northeast of Knoxville, on the Southern Railway. The district is centrally located in the Great Valley, or Appalachian Valley Province.

Prior to 1900, a small output of zinc was obtained from oxidized ore mined in open pits. In 1900, a shaft was sunk to sulfide ore, and in 1903, a second shaft was put down, neither being as deep as 200 ft. The American Zinc Co. of Tennessee began drilling in 1911, and outlined an orebody to which its No. 1 shaft was sunk, to a depth of 365 ft. Since that time, the American Zinc Co. of Tennessee has been the only operator in the district.

The Federal mining laws have never been in operation in Tennessee and mining lands are as irregular in boundary and area as farm lands. Ownerships are generally held in fee. In most cases the surface and mineral rights are transferred together.

Electric power is supplied at 66,000 volts by the Tennessee Power Co., from a hydro-electric plant at Parksville, Polk County, Tenn., 100 miles from Mascot. No other power is used, but an emergency steam plant is held ready in case of interruption of service.

Labor is all native and non-union. Negroes are employed for most of the shoveling, and for a few miscellaneous jobs, but for almost no other work.

Exploration

Exploration is carried ahead of mining largely by surface drilling. Both churn and diamond drills are used. Holes are not spotted according to geometric plan, but to block out ore outlines with the least number of holes. Churn-drill cuttings from each 5-ft. run are mounted for formation record, and in ore ground, all the bailings from each 3 ft. of advance are sampled and assayed. Diamond-drill cores are sampled and a specimen core filed for formation record. In ore ground, all the core

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† Mine Engineer, American Zinc Co. of Tennessee.

from each 5-ft. advance is ground and sampled for assaying. Shafts, drifts, and crosscuts are not laid out primarily for exploration, but grab samples are taken from the muck pile after each round is shot in ore for information on ore outlines. An underground diamond drill is also used, and sometimes drifts are driven to give position from which to drill.

No underground samples are taken, other than the grab samples from development work just mentioned; occasionally, cuttings from machine-drill holes are assayed to test for ore left above the roof of a stope. Cut samples would be tedious to take and would not be much more reliable than estimates because of the irregularity of mineralization. From the nature of the mineralization it is comparatively easy to estimate grade very closely on a freshly broken face.

Tonnage Estimates

Ore tonnages are estimated from assays and footages of surface drilling, supplemented by underground diamond-drill holes and development assays. Vertical drill holes represent prisms, the bases of which are polygons extending in each direction half way to the next hole. Tonnages and assays are calculated from areas, thicknesses, and average assays. A factor of 12 cu. ft. per ton is used. It is estimated that 15 per cent. of ore within the mine outlines is left as pillars for support of roof.

No statement can be made as to the accuracy of estimating methods. With improvements in mining and milling methods, lower grade ore can be mined and ore boundaries can be extended from time to time. The upper limit of mineralization, especially, is generally commercial, as are the lateral limits in part.

History of Former Mining Methods

Shrinkage stoping was tried in No. 1 mine, but was abandoned. Draw-off chutes were spaced about 30 ft. apart. Broken ore was drawn through loading chutes similar to the mill-hole chutes described later but with 36-in. gates instead of the 48-in. now in use. It was decided to abandon shrinkage stoping because of tendency of the rock to break in slabs too large to pass through the chutes.

Sublevel stopes, from which ore was trammed to raises and dumped to haulage levels below, were formerly used more than at present. In the future, for cleaning up after a mill hole or stope has been worked out, there will be a choice between sublevel stoping and slushing (or dragging). Probably no considerable tonnage will ever be produced from sublevels, however, because of the higher cost compared with methods now in use.

Geological Features

The general geological features of the district in which the Mascot deposits are located are given in various folios of the United States

Geological Survey. Mascot is located within the area covered by Folio 75. The rocks there show a strike varying from 50° to 75° east of north, and dip southeast at about 22°. Occasionally displacements are noted in the ground, generally measured in inches. The orebodies are about 1000 ft. vertically from the top of the Knox dolomite and follow that horizon fairly closely; the shape and size are shown in Figs. 1 and 2; Figs. 3 and 4 show the physical structure of the ore.

Most of the ore mined consists of dolomitic limestone seamed with veinlets of dolomite and sphalerite. Nodules and massive beds of chert

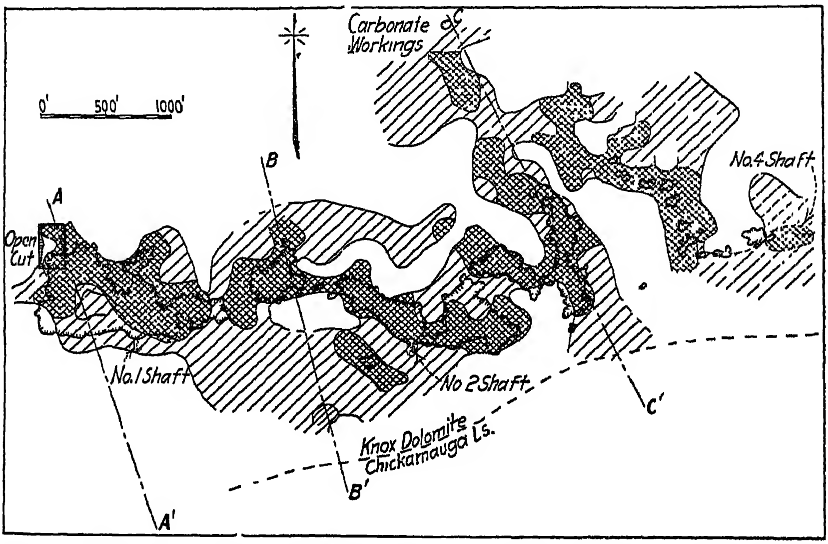

Fig. 1.—Plan of orebody, as shown by surface drilling and stope outlines of No. 1 and No. 2 mines. Single shading shows low-grade mineralization; double shading shows ore.

occur, especially under the ore horizon. Ground as much mineralized as that shown in Figs. 3 and 4 in most cases will not stand unsupported very long but the unmineralized limestone will stand unsupported, almost indefinitely, over widths up to 100 ft. if properly trimmed. Occasionally, thin shale partings between beds form planes of weakness and cause ground to fall. They are not common, however, and when occurring near the top of the ore, the overlying bed is usually carried as a roof. The minimum angle of gravity flow of the broken ore is a little over 45°; however, mill holes and stopes can be opened out to angles considerably flatter, on account of the way in which the drill holes are placed.

Mine Openings

At present, ore is hoisted from two shafts, No. 1 and No. 2. A third shaft, No. 4, is used as a ventilating shaft for No. 2 mine. Other open-

ings include three air shafts, 8 by 8 ft., and two abandoned hoisting shafts. Most of the mine tonnage at this time is hoisted from No. 2 shaft.

No. 2 shaft is 517 ft. deep to the main haulage level, and 582 ft. deep to floor of the skip-pocket loading station. A typical cross-section of the shaft is given in Fig. 5 and details of timber framing in Fig. 6. The

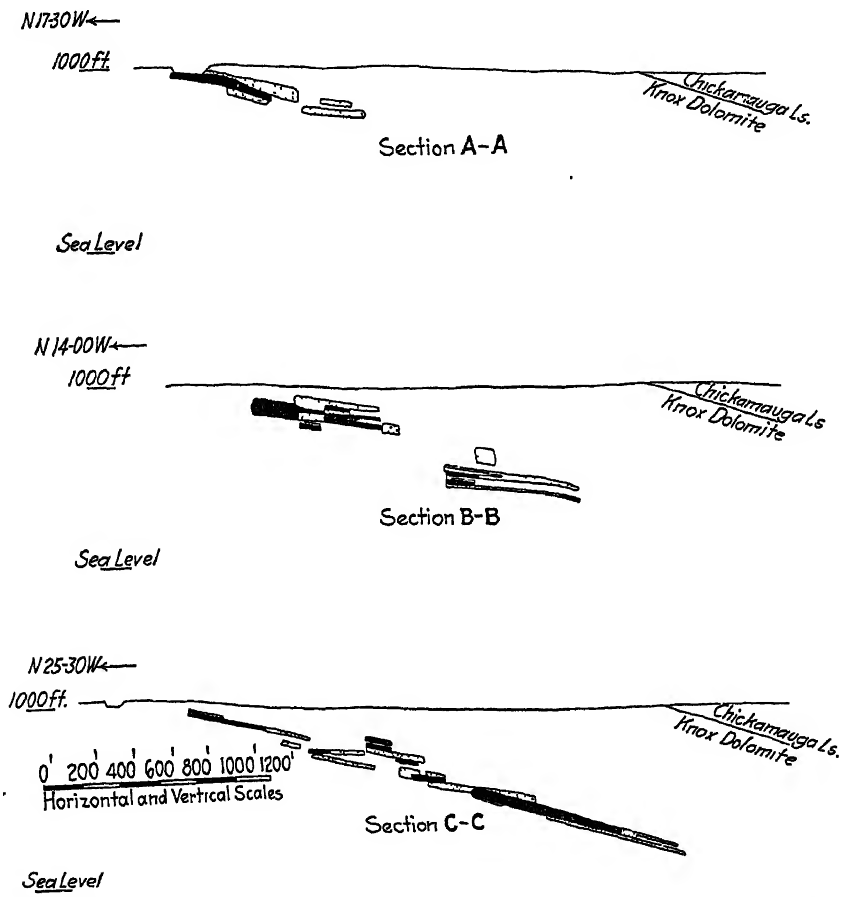

FIG. 2.—SECTIONS OF OREBODY AS SHOWN BY SURFACE DRILLINGS; SOLID BLACK INDICATES ORE.

shaft has been concreted through the surface clay, which has an average depth of about 48 ft. The timber sets serve only to carry the guides, and to carry pipes, electric power lines, and light and signal wires; the rock requires no timber supports.

A plan and section of orebody, as indicated by surface drill holes completed before No. 2 shaft was sunk, is shown in Fig. 7. The shaft went down in the hanging-wall side of the dolomite and bottomed near the middle of the orebody, and just outside its limits. A short drift

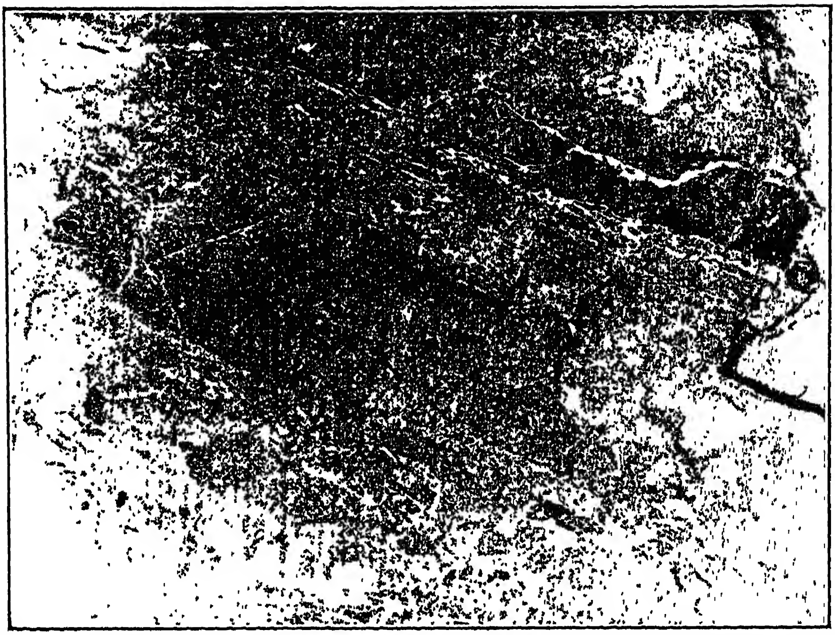

FIG. 3.—PLANE OF PHOTOGRAPH IS PERPENDICULAR TO STRIKE, HANDLE OF HAMMER IS PARALLEL TO DIP; IN VEINLETS BETWEEN ROCK FRAGMENTS CAN BE SEEN DOLOMITE (WHITE) AND SPHALERITE (LIGHT GRAY).

FIG. 4.—PHOTOGRAPH OF ORE; SPHALERITE IS LIGHT GRAY (IN PHOTOGRAPH) AND SHOWS SHINE ON CLEAVAGE FACES; DOLOMITE IS WHITE; AND KNOX DOLOMITE IS DARKER GRAY.

was driven to the ore and two short crosscuts in the ore, in the direction of the strike of the orebody. From those crosscuts, two inclines were driven, each about 600 ft. long. The one to the west was up 24°; that to the east was down 22°. Each followed closely the upper edge of the orebody. From each incline, drifts were turned off, at intervals varying

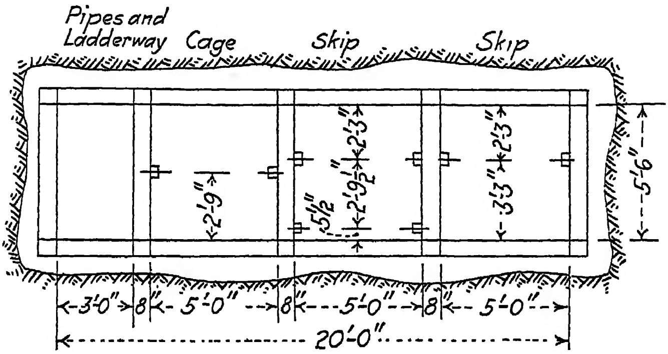

Fig. 5.—Section of shaft at No. 2 mine.

from 30 to 70 ft. vertically. From these drifts several stopes were opened up.

A comparison of Figs. 1 and 7 shows the desirability of completing surface drilling before locating shafts, also the large amount of drilling required to outline orebodies completely in this district. When No. 2

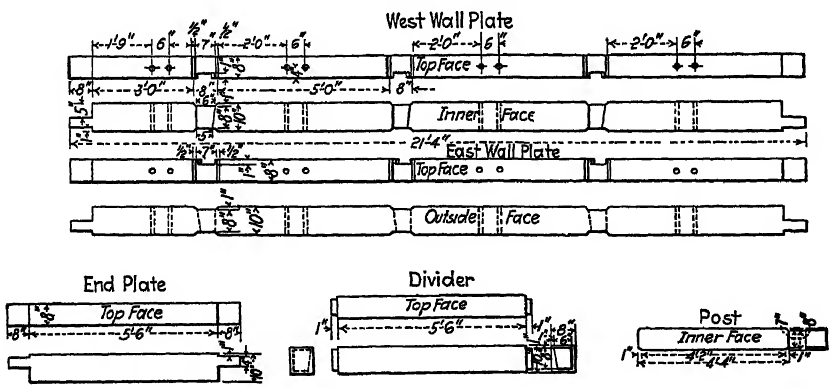

Fig. 6.—Shaft timbering at No. 2 mine.

shaft was begun, there had been completed about 65 surface prospect drill holes in the area shown on Fig. 7, or to the east of that. At present, there have been completed about 400 prospect drill holes in that area or eastward.

As a result of the small amount of prospect drilling that had been completed when No. 2 shaft was sunk, the development plans could

only be laid out for the ore in sight. The underground incline system was adopted as the best method of reaching the various mining levels. Fig. 8 shows the relative location of inclines to the orebody and also points where crosscuts were driven into the orebody for mining. The

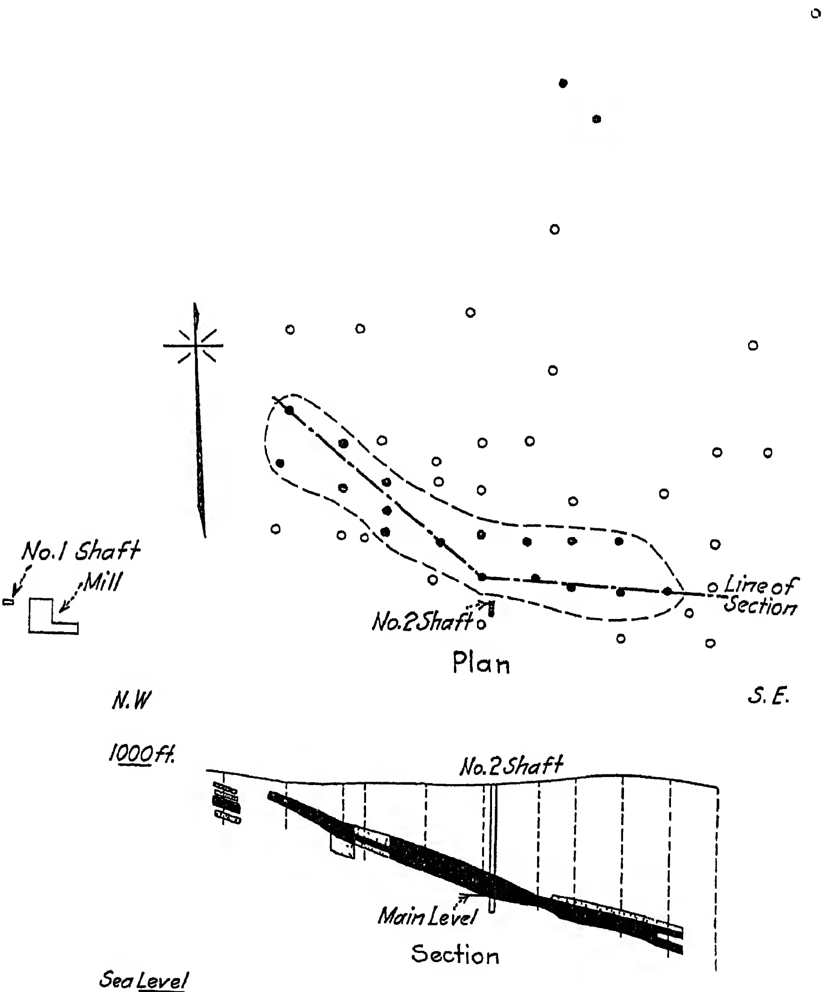

FIG. 7.—NO. 2 OREBODY AS OUTLINED BY SURFACE DRILLING AT TIME OF SINKING NO. 2 SHAFT; SOLID BLACK INDICATES ORE.

heading and bench, or underhand method of mining, was worked principally at No. 2. mine.

Development Drifts and Crosscuts

All development drifts are driven to size 8 by 8 ft. while crosscuts are driven 7 by 7 ft. The track grade in both drifts and crosscuts is held as near as possible to 0.5 per cent. in favor of the loads. Some curves in

main-level hand-shovel stopes have a radius of 15 ft.; but when possible, not less than 30-ft. radius curves should be used in stopes, and 50-ft. radius on main-line tracks. For motor haulage, 24-in. track gage of 40-lb. rail is standard, while a 20-lb. rail is used principally in stopes.

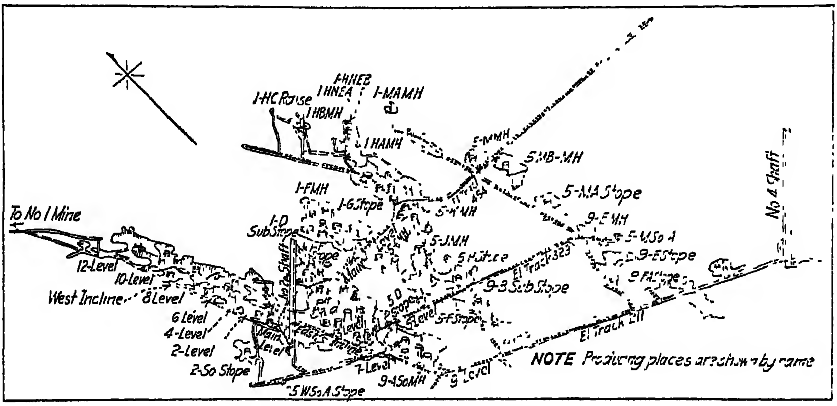

FIG. 8.—No. 2 MINE.

Drainage

All mine drainage is in open ditch along the track to the main sump, which is located at the shaft. Drainage on the lower levels is gathered into a single sump, where the water is relayed to the sump at the main-level pumping station.

Air Lines

All underground compressed-air lines lead from a 4 by 10-ft. receiver located near the shaft. The main line leaving this receiver is 6 in. but is reduced to 4 in. on the various levels and is again reduced to 2 in. in crosscuts. From these feed lines, 1-in. lines are taken off, at desired intervals, for supplying machines for both development work and stoping. Water lines are 2, 1, and ½ in. respectively, the 2-in. line being fed from a 10-in. pump discharge line in the main shaft. The size of water lines leading into the workings is governed by the number of machines to be supplied. At present, only the development machines and a few of the stope machines use water for drilling.

PRESENT MINING METHOD

Drilling and Blasting

Stope drilling and blasting under the present method of operating is done on the day shift, while the greater part of the development work, which involves drifting, raising, reaming, etc., is done on the night shift, the rock from the latter being shoveled on the day shift. Denver Dreadnought Model 60 hammer drills, mounted and unmounted, are standard for stope work, and Ingersoll-Leyner Model 248 drills are used for drift-

ing. All blocking in stopes and mill-hole chutes is done with Ingersoll-Leyner BCR 430 Jackhamers. Hollow round 1¼-in. steel with Carr bit is used entirely in drifting and stoping, while in raising 1-in. cruciform steel with cross bit is standard. All blocking and reaming machines use ⅞-in. hexagon steel with Carr bit.

Drill Rounds

The method of placing drill rounds in stopes and drifts is shown in Figs. 9 and 10. About the same method of placing holes in raises is used as in drift heading. Explosive used for all classes of work is 30 per cent. low-freezing gelatin dynamite, 1⅛ by 8 in. sticks (extra plastic).

Mill-hole Mining

Development plans were altered, in 1920, to conform with the mill-hole or glory-hole method of mining, which had been tried and was

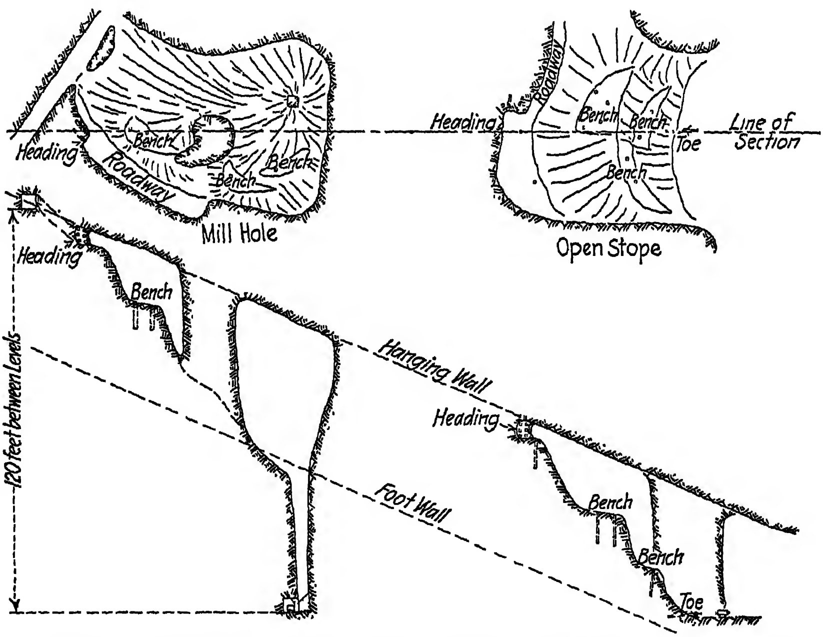

FIG. 9.—PLAN AND SECTION OF No. 2 MINE, SHOWING METHODS OF PLACING HOLES.

adopted at that time. A plan and section through a typical mill hole and stope are shown in Fig. 9. Where it is possible, the main-level haulage drifts are driven in the foot wall and raises are driven through the orebody at intervals where the greatest tonnage is available to each raise. These raises are connected with manway drifts, or crosscuts, driven in the hanging wall. The heading and bench method of mining is worked. Headings 8 ft. in height are cut around the raise in four directions between pillars that have been located as far away from

the center of the raise as the roof conditions will permit; after the headings are formed and pillars are located the bench is mined out by the underhand method.

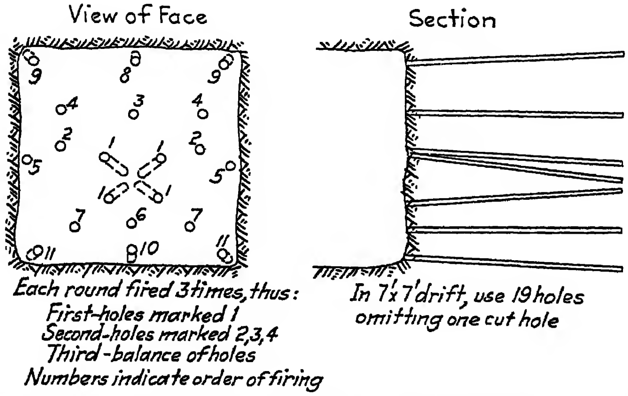

FIG. 10.—STANDARD 20-HOLE DRIFT ROUND FOR 8 BY 8 FT. DRIFTS.

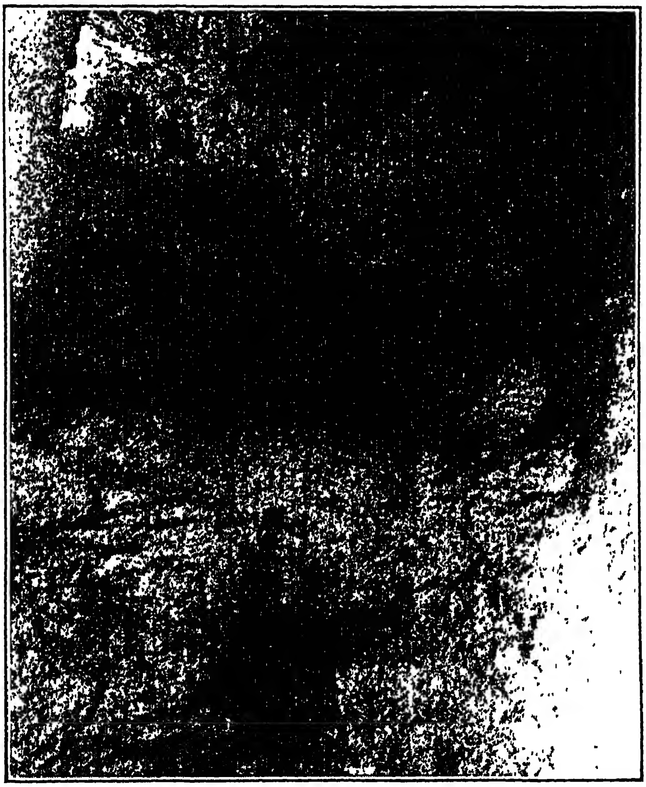

FIG. 11.—MILL HOLE, SHOWING HEADING AND BENCHES.

All raises for mill-hole mining are driven 5 by 5 ft. and later are reamed to 10 by 10 ft. The ore is drawn off through 48-in. steel chutes

on the main haulage level. (See Fig. 12.) Tramming from chutes to lay-by is by hand. Trolley and storage-battery locomotives haul cars to the tipple or incline in trips of five to ten cars each.

When planning development work for the mill-hole mining system, the available tonnage to each raise is figured and an effort is made to hold the development cost within 5 to 7 cents per ton. The relation of some of the mining levels to the orebodies makes the cost of development

FIG. 12.—STEEL MILL-HOLE CHUTE.

prohibitive for mill-hole mining; therefore, it is necessary to work a number of hand-shovel stopes. Such stopes are opened up from the hanging-wall side; the heading and bench method of mining is worked as in the mill-hole system.

The following percentage figures for the year 1923 show production from the two methods of mining, together with the tonnage secured by development and by the contract system as a whole; the latter is explained later.

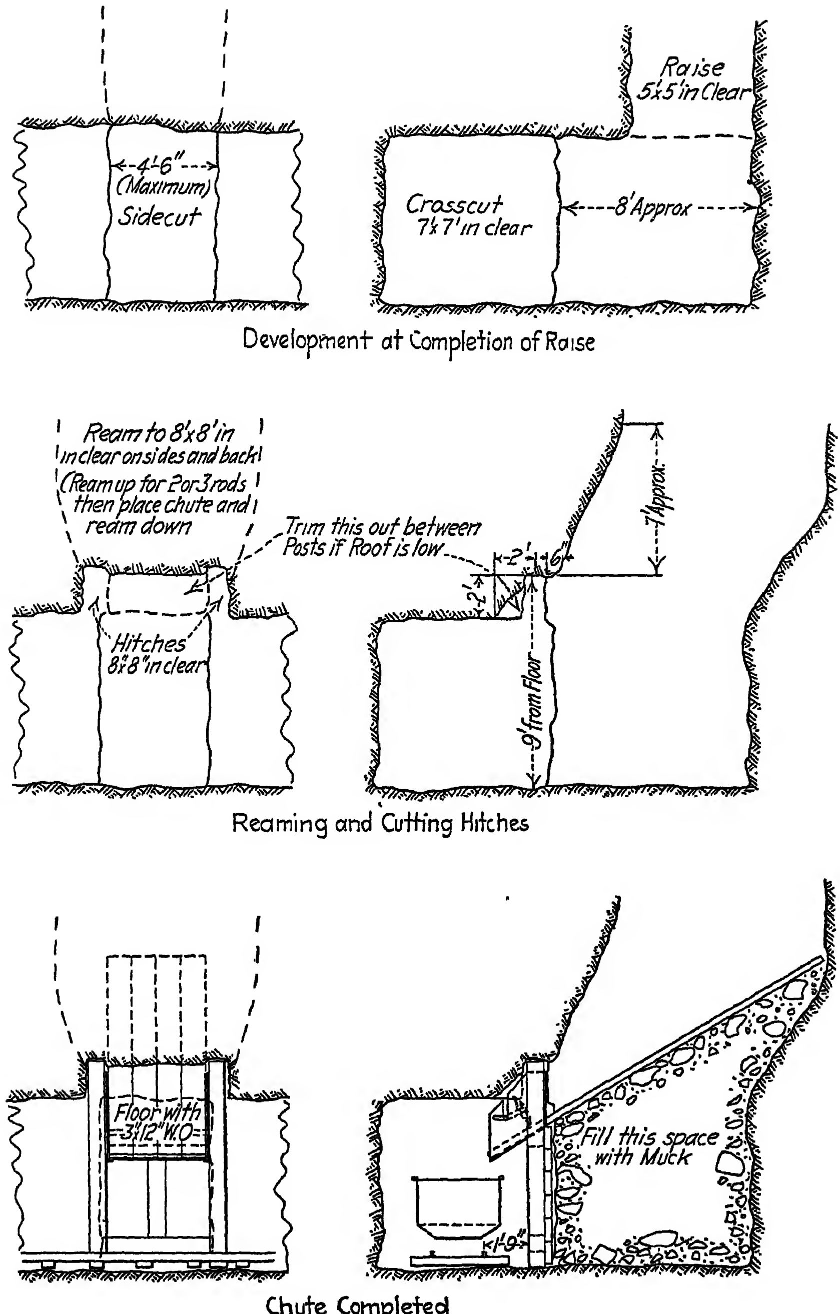

Fig. 13.—Standard steel mill-hole chute, development details and assembly.

No. 2 Mascot, American Zinc Co. of Tennessee, 1923

	Per Cent.
Produced by mill-hole method of mining	52.11
Produced by open-stope method of mining	35.19
Produced by development	3.36
Produced by contract mining (under above methods)	90.66
Produced by company time work	9.29
Skip pocket and bin adjustment for year 1923	0.05
Total	100.00

The comparative cost figures on mill-hole and open-stope method of mining given herewith for 1923 are based on contract tonnages only:

	Tonnage	Cases Powder	Tons per Case	Daily Wage Contractor
Mill hole	185,282	1284	144.30	$6.05
Open stopes	124,592	724	172.08	7.67

Cost per Ton

	Powder, Cents	Labor, Cents	Contractor, Cents	Total, Cents
Mill holes	6.16	12.25	5.71	24.13
Open stopes	5.20	3.51	8.96	17.66

The difference of 8.74 cents per ton in labor cost between the two methods is because the costs under the mill-hole method include the labor for loading and tramming, while to the open-stope method there must be added the hand loading cost of approximately 13 cents. This makes the total cost of this ore in cars 30.66 cents, as against 24.13 cents under the mill-hole system.

Contract Methods

Under the contract and mill-hole system, the production per total man underground has increased from 4.45 to 8.50 tons. The contractors in the mill-hole stopes furnish all labor, explosives, and wrenches, and draw the ore out of chutes into cars and tram it to nearest lay-by. They also do all blocking and are required to keep the roof in safe condition. The company pays all labor at its regular payday, and charges same to contractors. All tonnage is figured by a car factor arrived at by the mill weigher, located at No. 1 mill, and is subject to change each week. With this method of calculating tonnage it is possible to secure full cars at all times.

Under the open-stope contract method, the contractors furnish labor, explosives, and wrenches, do all blocking for shovelers, and assist them in moving and placing shoveling boards. The contractors are also required to keep the roof in safe condition.

All contracts, other than development drifts and raises, are let for a period of from 3 to 6 months; regular forms are made out and contracts are not in force until properly signed and witnessed. When figuring the contract price for each individual stope or mill hole, the physical conditions are taken into consideration; viz., the accessibility of the stope to the blacksmith shop, the thickness of ore to be mined, the length of tram, etc., all of which necessitates a variation in prices. A sliding scale is used when opening up stopes from the tight; this scale is such that from 4 to 6 months after starting, the price will have reached the average low rate set for each working place.

Loading and Haulage

Hand Loading

All hand loading is done by colored shovelers who work on a contract task basis. They are paid 23 cents per car (averaging 1.7 tons each), but they are not allowed to leave the mine until they have loaded 13 cars, unless they secure a cage pass from their foreman. An average of 16.80 tons per hand shoveler was obtained for the first 6 months of this year.

Mules are used for gathering cars in hand-shoveled stopes. Where two or more pair of shovelers are working in one stope, a driver is furnished; but for only one pair, the shovelers do their own driving.

Mechanical Loader

Several types of loading machines have been tried out since 1916, also several attempts made to slush the ore into raises or cars; but as the ore breaks very large only the heavy type of loading machines were found at all satisfactory. The two types of machines giving the best results were the No. 0 Mine type, electrically driven Thew shovel, and a Model 25 Marion steam shovel that was converted from steam to air. Since adopting the mill-hole method of mining, mechanical loading has been discarded.

Haulage

Until this past year, storage-battery and gasoline locomotives were used entirely for haulage, mules being used for gathering. The storage-battery locomotives used were 4-ton General Electric and Jeffrey; all were equipped with 63-A-6 Edison cells, two sets being used for each locomotive. Milwaukee 4¼-ton gasoline locomotives were used very

satisfactorily on long hauls where ventilation was good, but on account of the gases no attempt was made to use them in tight drifts or stopes. Increasing long hauls have necessitated larger equipment and the 4-ton locomotives are being replaced with 6-ton Jeffrey locomotives, for the lower level haulage, while General Electric trolley locomotives have been installed on the main level, the trolley locomotives being equipped with gathering reels. The 6-ton storage-battery locomotives are equipped with 70-A-12 Edison cells, using two batteries to each locomotive.

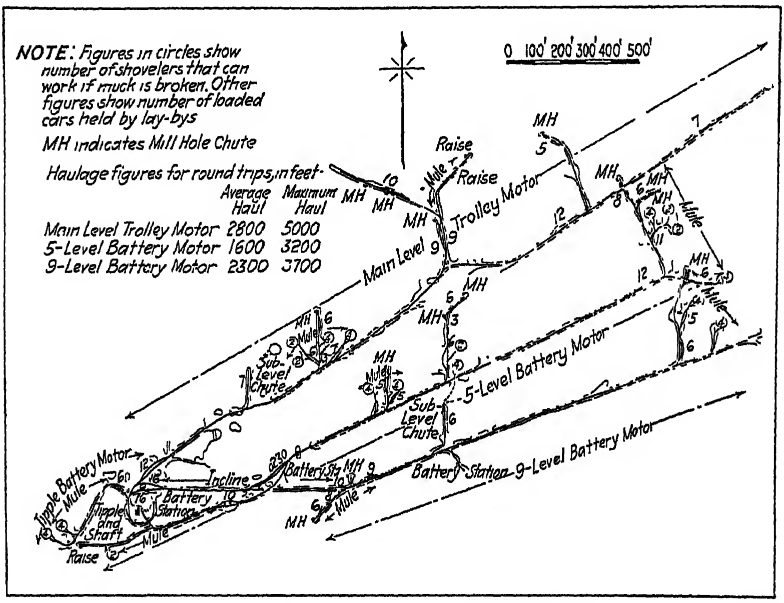

FIG. 14.—HAULAGE SYSTEM NO. 2 MINE.

Motor-repair stations and battery-charging leads are maintained on all levels. A motor generator and charging panel with voltage regulator is located at the main level station.

Storage

The necessity of seeing the working face after each shot, in order to maintain the desired grade, makes it practically impossible to store any ore in the stopes. Under the present method of mining, all open stopes are cleaned each day, while the ore left in the mill holes on the day shift is drawn out on the night shift. A skip pocket of 750-ton capacity, located at the shaft, gives sufficient storage not to delay tramming should a breakdown of the hoisting equipment occur. All ore is dumped into this pocket through a two-car rotary tipple, electrically driven, the ore being drawn out of pocket through steel chutes directly into skips; the

flow is controlled by fingers operated by air cylinders suspended above. The skip-pocket chutes and method of controlling skip loading are shown in Fig. 15.

Both skips and cage hoists operate in balance. Skip hoist is of the tight-drum conical type (single reduction). Skips work in balance, one

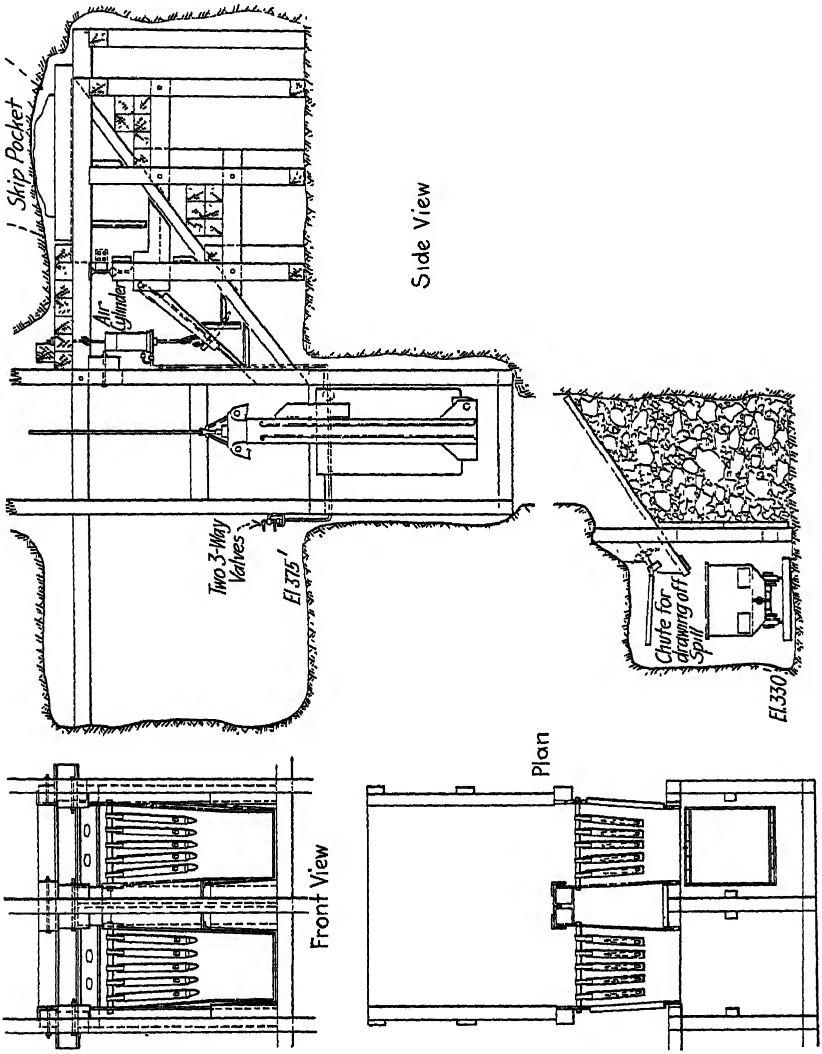

FIG. 15.—SKIP-POCKET CHUTE AT NO. 2 MINE.

being in loading position in the skip pocket, while the other is in dumping position in the headframe. The cage hoist is counterbalanced by a weight traveling in the center compartment. Both skip and cage are equipped with safety dogs, which are tested once each week. Overwinding switches are located at the top of the head frame; these will cut

off the power from the hoists, but it is necessary to apply the brake by hand. The hoisting speed, both cage and skips, is 500 ft. per min. Hoisting ropes now in use on the skips are American Steel & Wire, 1¼ in. 6 × 19, and 1⅛ in. 6 × 19 on the cage and counterbalance.

Water Protection and Pumping

A great deal of water was encountered during the early mining, the source of which was later traced to the bed of a creek that crosses the

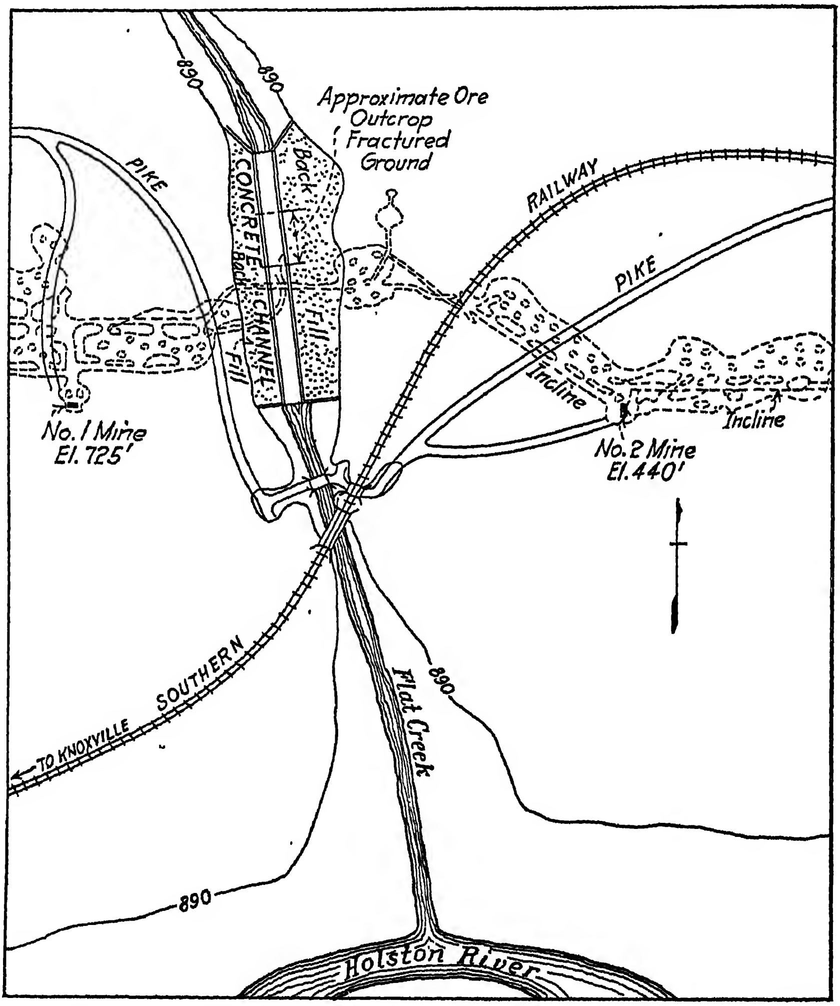

FIG. 16.—RELATION OF CONCRETE CHANNEL TO MINE WORKINGS.

main ore outcrop between No. 1 and No. 2 mines. A concrete channel, 800 ft. long, was built to carry the creek over this area. The relation

of the concrete channel to the mine workings is shown in Fig. 16 and the detailed construction of the channel walls in Fig. 17. Since this channel was completed there has been little water trouble. The normal flow during the summer months is approximately 1000 gal. per min.; during the rainy season and sometimes during the winter months the flow is much greater.

The pumping equipment at No. 2 mine consists of one 450-gal. Gould triplex pump and four 1000-gal. Jeanesville centrifugal pumps, all pump-

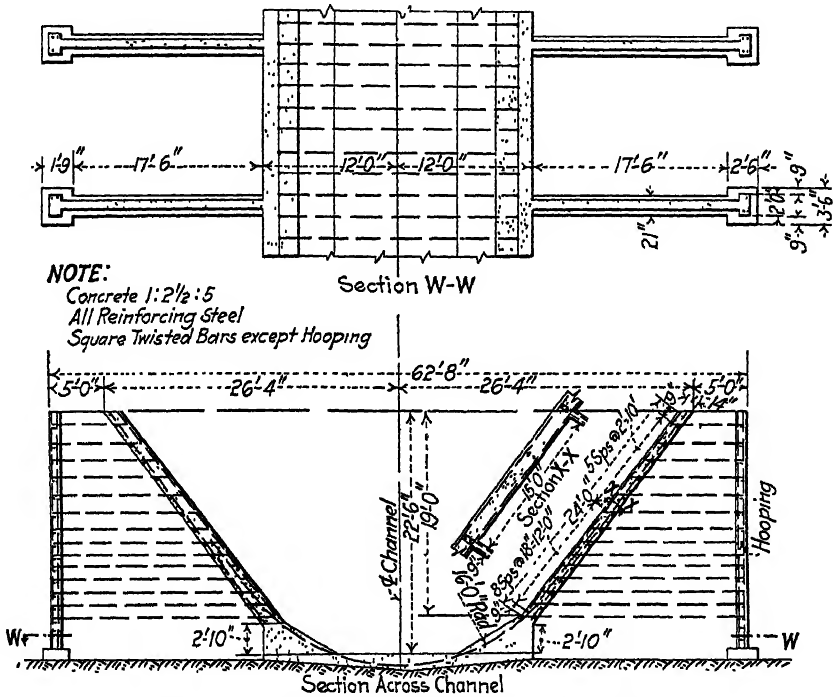

FIG. 17.—DETAILS OF CONCRETE CHANNEL WALLS.

ing directly to the surface, a distance of 520 ft. No. 1 mine pumping equipment consists of one 750-gal. Prescott duplex pump, two 1000-gal. centrifugal pumps, and two 2000-gal. Layne & Bowler emergency pumps. The latter are located in a special pumping shaft near No. 1 hoisting shaft. The total pumping capacity at both mines is 11,200 gal. As an extra protection, concrete-steel bulkheads have been constructed in various drifts and also on main level entrance to shaft where pumps are located; see Fig. 18. These can be closed in case of flood, giving protection to all pumps.

Compressor Plant

A central compressor plant, located at No. 1 shaft, furnishes air to both mines. The equipment consists of one Nordberg 3400-cu. ft. compressor and two Ingersoll-Rand 1750-ft. compressors, making a total capacity of 6900 cu. ft. The three compressors are electrically driven

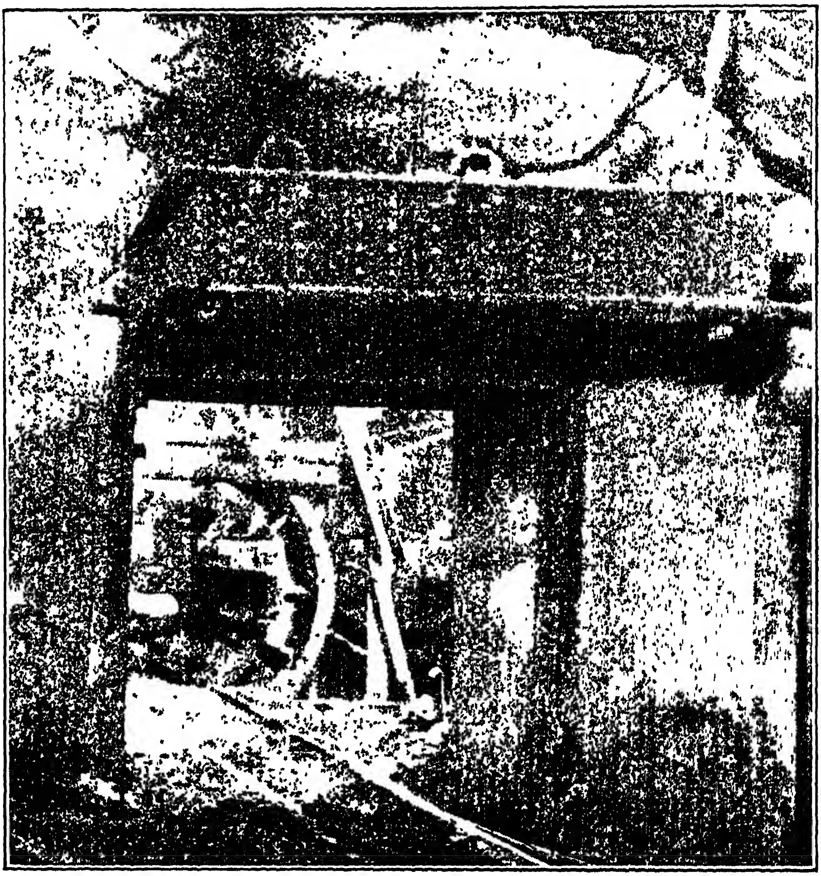

Fig. 18.—Bulkhead at No. 2 mine main level between main-level stations and mine workings. To close, track is removed and loose muck cleaned from floor; door is then raised by chain falls and unhooked. Through opening may be seen rotary tipple, which dumps two cars at a time.

by 2300-volt synchronous motors. The air line feeding No. 2 mine is approximately 2800 ft. in length and is on surface following the contour of the ground. This is an 8-in. line that enters a receiver at the shaft; from the receiver a 6-in. line enters the mine through the pipe compartment in the shaft.

Ventilation and Lighting

One 45,000 cu. ft. Buffalo Forge conoidal fan, located on the surface at No. 4 shaft, ventilates the entire mine. This shaft is connected to the east incline in No. 2 mine by a drift. Stope and raise connections are made from the various workings and levels above, which gives the proper circulation of air through most of the working places. This fan is an upcast, which makes the main hoisting shaft a downcast. The intake and outlet of the fan are so arranged that opening one door on the intake side

FIG. 19.—MAIN-LEVEL LAY-BY, NO. 2 MINE.

and closing a door on the outlet side reverses the direction of the air in the hoisting shaft. This change is only made at times during the winter months to keep the skips from freezing up in the shaft.

The shaft station, inclines, and haulageways are electrically lighted. The workmen are required to furnish their carbide lights but the company furnishes all carbide.

Disposition of Ore

The ore, on reaching the surface, is automatically dumped into a 50-ton bin in the head frame, where it is fed by a pan conveyor into a gyratory crusher, then to a Symons disk crusher, from which it is elevated to a 300-ton bin. From this bin the ore is transferred to a bin at No. 1 mill by aerial tram, 2800 ft. in length.

Distribution of Labor and Costs

The following table shows the distribution of labor at No. 2 mine, which is producing at this time approximately 1400 tons per day of two 9-hr. shifts. Of this amount, approximately 1000 tons are trammed on the day shift and 400 tons on the night shift.

Labor Distribution

	Day Shift	Night Shift
Foremen	1	
Shift bosses	2	1
Pumpmen	2	1
Machine men, operating	24	
Machine men, development	4	6
Machine helpers, operating	8	
Machine helpers, development	2	2
Blacksmiths	2	
Timbermen	1	1
Incline couplers	4	3
Trackmen	3	
Car repairers	2	
Motor couplers	4	2
Incline hoistmen	1	1
Mule drivers	3	1
Motormen	4	3
Tipplemen	1	1
Skip loaders	2	2
Hoistmen	2	1
Cage tenders	1	
Tool boy	1	
Change house	1	
Motor repairmen	1	
Miscellaneous	8	2
Diamond drillers	2	
Pipe men	1	
Shovelers	24	6
Chute pullers	14	10
Total shifts	125	43

The average mining cost at No. 2 mine for the first half of this year is 78.37 cents per ton, which includes 13.28 cents deferred development and 3.63 cents miscellaneous deferred charges.

Distribution of these costs, together with the physical data for this period, is as follows:

MINING COSTS

		Total Cost, Cents per Ton	
Drilling		17.63	
Blasting, total	9.04		
Blasting, development	3.10		
Blasting, less development		5.94	
Drill boulder breaking		0.50	
Total breaking ground			24.07
Roof protection			0.06
Timbering			0.69
Loading			12.91
Level haulage		7.15	
Incline haulage		2.13	
Track		1.25	
Ore cars		1.64	
Total haulage and car repairing			12.17
Transfer at shaft			0.96
Hoisting and skip loading			4.28
Ventilation			0.09
Sundry and development			23.14
Total cost, mining			78.37
Less development			13.28
Mining cost exclusive of development			65.09

During this period a total of 2562.5 ft. of development work was done in No. 2 mine, which is covered by the above deferred cost, 13.28 cents per ton on the total tonnage hoisted. The development work involves drifting, raising, reaming, and slabbing of lay-byes.

PHYSICAL DATA

Total tons hoisted	212,990
Tons per shift (2 shifts per day)	696.05
Skips hoisted per shift	175.93
Tons per skip	3.96
Tons per mine car	1.71
Drilling	
Number of machine shifts	3,167
Number of holes drilled per shift	66.16
Number of holes drilled per machine shift	6.39
Number of feet drilled per machine shift	53.99
Tons per machine shift	67.25
Material and Averages	
Loading powder, cases	2,414
Tons per case powder (50 lb.)	88.23
Feet drill steel	964
Carbide, pounds	14,800
Feet air hose	250

Safety and Welfare Work

Safety and welfare work begins with a physical examination of all employment applicants. Safety propaganda is conducted by aid of bulletins and display posters furnished by the National Safety Council and other organizations.

Inspections of safety conditions are made by the following: Surface operations, by a workmen's safety committee, composed of one man each from the mechanical department, the limestone plant, and the mill; underground operations, by mine foremen and shift bosses; all operations, by inspectors of a liability insurance company; underground operations, by state mine inspector. All these make detailed reports and recommendations. The reports are discussed and acted upon at monthly meetings of a general safety committee, consisting of five department heads.

An emergency hospital is maintained, and a doctor and visiting nurse are in regular attendance. In addition, full first-aid cabinets, stretchers, blankets, etc., are kept in department offices and in each mine. Minor first-aid treatment is administered underground by tool boys, who have experience in that work.

Caving Methods

IN A straight caving system, the ore is first undercut and then broken down by its own weight or by the weight of the overlying rock, or by a combination of both. Operations that involve the caving of the material overlying an orebody as an essential part of the work are also classed as caving, though practically all the ore is broken by drilling and blasting. Actually there are two distinct methods—sublevel caving and block caving, and modifications thereof. The mines described are Miami and Alaska Juneau.

Mining Methods of the Miami Copper Co.

By J. H. Hensley, Jr., Miami, Ariz.

(New York Meeting, February, 1923)

The mine of the Miami Copper Co. is in the Miami district, Gila County, Ariz., approximately 7 miles west of Globe. In 1906, the General Development Co. secured the ground now owned by the Miami Copper Co., and about a year later discovered ore in a prospect shaft at a depth of 220 ft. The Miami Copper Co. was organized in 1907, the claims held by the General Development Co. were transferred to it, and an active development campaign was inaugurated. The railroad was extended from Globe to Miami in 1909, a concentrating plant was constructed, and the company began producing early in 1911. The unit size of mineral tract in the district is the standard lode claim 600 ft. in width by 1500 ft. in length. Ownerships of land are held in fee.

The geology of the district has been described by F. L. Ransome.[1] The formations of local importance are: The pre-Cambrian Pinal schist, which consists mainly of metamorphosed sedimentary rocks and is commonly of a thinly laminated sericitic variety; the diabase, which is thought to have been intruded in late Paleozoic or early Mesozoic times; the Schultze granite and granite porphyry, considered early Tertiary; the dacite of late Tertiary time; and the Gila conglomerate of early Quaternary.

General Items Influencing Mining Methods

The bodies of disseminated ore in the district are flat lying masses of irregular outline and variable thickness. As a rule, these masses lack definite boundaries; the orebody follows a curve along one of the main ridges, approximately 5500 ft. in length with a maximum width of 1600 ft. The minerals occur as finely divided and uniformly disseminated chalcocite in the Pinal schist and Schultze granite. The typical overburden or capping is a barren rusty brown, very friable, highly siliceous, leached schist, or granite-porphyry.

The topography of the district is fairly rugged and irregular; the altitude ranges from 3400 ft. above sea level, at the town of Miami, to 3900 ft. on the hills above the Miami orebody. The climate is mild; the mean annual temperature is 63.7°F.

[1] U. S. Geol. Surv. Prof. Paper No. 115.

A plentiful supply of water is available, by pumping from a depth of 200 ft., in the main valley.

The pumping problems may be disregarded in the district; the Miami mine makes an average of 75 gal. of water per minute.

Oil is the chief fuel; it is obtained from California, Mid-Continent, and Mexican fields.

The main source of timber is Washington and Oregon for sawed stock and Texas and New Mexico for round stulls and lagging; little Arizona timber is used.

Explosives are obtained from the plant near Benson, Ariz.; all supplies are shipped in by rail.

Underground labor is from 60 to 75 per cent. Mexican. About half of the remainder is native American and the other half is made up of the various European races. Except the Mexicans, the efficiency of the underground labor compares favorably with other districts. Practically the entire district is operated on the open-shop plan; nearly all mechanics, engineers, etc. are union members, while the miners are not strongly organized. The district is easily accessible both by rail and highways.

Exploration has been carried on by shaft sinking, drifting, raising, churn drilling, and diamond drilling.

Sampling

Shafts, raises, and winzes are sampled by taking the cuttings from horizontal shallow channels 1 to 2 in. in depth by 4 to 5 in. in width cut at intervals of 5 ft. Drifts are sampled by horizontal channels on each side approximately 4 ft. above the floor; the cuttings from 10 ft. of channel, 5 ft. on each side of the drift, are combined and represent the sample for that 5 ft. of drift. No great care is taken in cutting these sample channels, as experimental sampling indicates that no greater accuracy is obtained and considerably more time is required. No consideration is taken of the structure of the ground or the presence of visible enriched veinlets. In churn drilling, samples were cut from the sludge, produced by each 5 ft. of hole drilled. In diamond drilling, all of the sludge for each 5 ft. of hole was taken as sample; as about 15 per cent. of the hole was cored, cores were not assayed except in special instances.

Estimating Tonnage and Grade

In contrast to the general practice in estimating low-grade orebodies, the estimates were based almost entirely on underground drifting and raising development. Churn-drilling was largely used to delimit the lateral extensions, and diamond drilling to determine the depth of the orebodies. Estimates are based on the allowance of 12½ cu. ft. of ore in place to 1 ton of ore.

There are three grades of ore: (1) Sulfide ore, containing 1.5 per cent. or more copper with not more than 0.6 per cent. copper in the form of oxides and carbonates; (2) oxide ore, containing 1.5 per cent. or more of copper with 0.6 per cent. or more copper in the form of oxides or carbonates; and (3) low-grade sulfide ore, which contains more than 1 per cent. and less than 1.5 per cent. copper.

The figures on which results of mining operations are based are obtained by tonnage and assay values calculated from development by drifts, supplemented by results of churn and diamond drilling. To date, the operations of this company have been confined to the extraction of the sulfide ore.

In the undercut caving system of mining, tonnage estimates are made from vertical sections, parallel to the direction of the drawing operation, taken at 25-ft. intervals; the ore limits on these sections are obtained from the sampling of the final drift and raise development, of which there is an average of 1 ft. to each 47 tons of ore in place. The same factor, 12½ cu. ft. per ton, is used in these calculations.

The assay value of the ore in place is based on the average assay value of parallel drifts only, an average of one 5-ft. drift sample for every 238 tons of ore in place. The assay value of the ore in place, as calculated from the drift samples, is reduced 10 per cent., and this figure is used as the value of the ore in place. This reduction of 10 per cent. in assay was based on the comparison of results obtained from the channel drift samples of approximately 600,000 tons of ore, which were extracted without dilution of waste rock by the top-slice system of mining, with the actual copper content of the ore as determined by milling operations.

Mining Methods Formerly Used

The mining methods formerly in use at the Miami mine were as follows:

Square-set System

This system was employed at the beginning of operations to mine the peaks of the orebody, which were irregular in outline and relatively high grade in value. The extraction by this method served to form a timber mat that separated the ore and barren capping over that section which was later mined by the shrinkage system. This method also formed an even boundary in another section, from which extraction by means of a horizontal top-slicing system could be advantageously inaugurated.

Shrinkage Stoping

This method, the distinguishing feature of which was the use of rooms and pillars, each 50 ft. in width, was well adapted to the section of the

mine in which it was used. The ore was a hard, strong schist of an average copper content of 2.7 per cent. A complete description of this system was published in Vol. 55 of the *Transactions.*

Top-slicing Method

Following the extraction of the peaks of the main sulfide orebody by square setting, mining was carried on by a top-slicing method, in which an attempt was made to work slicing faces varying from fifty to several hundred feet in length and retreat toward a transfer shaft outside of the slicing area. Supplies were hoisted from the haulage level to the slice levels through this transfer shaft by means of a 5 by 10-ft. single-deck cage, operated by a 75-hp. electric motor. Entrance for supplies and men to the working faces was had through long drifts from the transfer shaft. It was very difficult to keep these connecting drifts open and extensive repairs were necessary, which greatly interfered with operations and resulted in excessive mining costs. The control of operations was difficult or nearly impossible as the various working faces advanced irregularly. A block method of slicing was then devised, wherein the blocks, 250 ft. square, were operated as independent units. Each block was served by an independent supply raise at its center; a much greater proportion of productive area was made available and the number of active working faces in a given area greatly increased. A description of this method was published in Vol. 55 of the *Transactions.* This system was replaced by the undercut caving system now in use.

Shrinkage Stoping

Shrinkage stoping was applied to a soft, weak, much fractured ore, in which stopes averaging 8 ft. in width were driven at intervals of 25 ft.; the pillars between the stopes were later undercut and were broken by caving. The undercut caving system was developed from this system; and after being applied to the mining of a separate orebody of 3,800,000 tons, it was decided to abandon the top-slicing method in favor of it. The more important reasons for the abandonment of the top-slicing method in favor of the caving method may be briefly summarized as follows:

Advantages of Caving System

1. The net profit to be obtained from caving the orebody was greatly in excess of that possible by slicing it.

2. The number of men required to maintain a fixed production was greatly reduced and a smaller proportion of skilled men was required.

3. The number of accidents per 1000 tons of ore mined was greatly reduced.

4. Fire risk was lessened.

5. Ventilation was more easily controlled and better ventilation was possible than with the slicing system.

6. Production is much more elastic and easily controlled. Preliminary development and undercutting operations may be carried on well ahead of requirements and we have been able to maintain the required production over periods of acute labor shortage by transferring available men from other work to drawing operations. The daily tonnage hoisted may easily be kept at a fixed amount which, from the relatively small storage capacity for broken ore both on surface and underground, is necessary for the operation of the concentration plant at a uniform and economical rate. This would be impossible if the entire production of the mine was obtained from slicing operations, as then the daily output depended directly on the total number of men at work on that day; the absence of 10 per cent. of the crew resulted in 10 per cent. less production.

The daily tonnage that could be obtained by working a block of the orebody of a limited area was in favor of the undercut caving. The block system of top-slicing then in use, in which the slices were 10 ft. in height and 250 by 250 ft. in area, yielded an average of 877 tons per day for the life of the slice, or 71,265 sq. ft. of area of orebody was required per 1000 tons daily production. In the caving system this figure will vary according to the physical character of the ore caved, on the weight exerted by the overburden, and on the height of ore caved. The average production from caving in 75-ft. lifts is approximately 1000 tons daily from an active drawing area of 11,337 sq. ft., but from existing conditions and operating details an allowance of 50,000 sq. ft. per 1000 tons daily production is made.

The only important factor favoring the slicing system was the recovery of practically 100 per cent. of the ore with no appreciable admixture of barren capping. Based on costs of slicing and caving in this mine, it was calculated that the caving system would prove more profitable than the slicing system, provided at least 32 per cent. of the tonnage was recovered clean, assuming that as drawing proceeded the grade of ore after dilution started would decrease uniformly to a point where the product would be barren capping. At this time, 2,577,734 tons of ore assaying 1.79 per cent. copper had been drawn from a section estimated to contain 2,319,627 tons of ore assaying 1.986 per cent. copper, showing a tonnage extraction of 111.13 per cent. and a copper extraction of 100.16 per cent. The extraction of over 100 per cent. of the copper is accounted for by the fact that the capping contained an appreciable amount of copper of which no account was taken. The "grade extraction" or ratio of the grade of the ore drawn to the grade of the ore in place was $\frac{1.79}{1.986} = 90.1$ per cent.; and it was estimated that 80 per cent. of the total tonnage was drawn clean before any dilution appeared. As the results obtained from this

section were so greatly in excess of what was required to make this system more profitable than the slicing system, the argument based on cleaner ore extraction in favor of the slicing system lost most of its weight.

Mining Methods Now in Use

The undercut caving system of mining is now applied to a section of the main orebody approximately 700 by 1000 ft. in extent, in shape very irregular. A number of sections through the orebody show the average thickness of ore to be 215 ft. and capping 357 ft.; a typical section through the center of the orebody shows 400 ft. of ore and 350 ft. of capping. The orebody is limited above by a very irregular contact with a barren capping, which is a rusty brown, very friable, leached schist, or granite-porphyry. It is easily distinguished by its color, but difficult to control as it breaks readily into fine particles and does not pack; it flows more easily than the ore. The boundaries of the orebody are very irregular. On the top it is bounded by barren or oxidized capping; on the bottom, by enriched protore; on the sides, by low-grade ore or oxide ore of varying grade; and in some cases is sharply faulted against barren rock. In some cases the orebody is bounded by a vertical plane representing the property line. There is a perceptible dip of the orebody to the east, but it is so irregular that when taken into account with the extent of the orebody it would have no material effect in determining the mining system used; although in the case of undercut caving, it is one of the elements determining the vertical extent of each lift.

The Ore

The sulfide ore is a highly altered Pinal schist or granite-porphyry of a very light-colored groundmass, darkened to a pale gray by the occurrence of chalcocite in small particles and irregular veinlets. The texture is fine grained with numerous small quartz stringers traversing the rock in all directions. The gangue has been converted to quartz and sericite. The hardness of the ore varies from ore that can be drilled with a standard air-feed stoper drill at the rate of 34 in. per min. to ore in which the drilling speed is 3.5 in. per min. with a Waugh turbro drill. The average hardness is about the mean of these limits—that is, what would be considered a soft rock, easily broken, and very friable. The friability of the ore is well shown by the fact that 85 per cent. of the mine production as delivered to the coarse crushing plant passes through a 2-in. bar grizzly. The average moisture content of the ore is 3.1 per cent.

The average grade of this orebody is 2.26 per cent. copper; practically all of the copper is in the form of chalcocite. The mineral is uniformly distributed over large masses. Waste or low-grade zones are of such extent that they may be mined with the ore without seriously

affecting the grade of the ore drawn or they may be left without affecting the mining methods in use.

The minimum angle of gravity flow of broken ore through timbered and untimbered raises or ore passes is 53°.

Water, Temperature, and Gases

The presence of water in the softer grade of ore, even in very small quantities, is objectionable as it greatly increases the tendency of the ore to pack, not only in the transfer raises but in the caving operations. Caving operations, or rather the drawing of the undercut ore, is greatly facilitated by opening any section a month or so in advance of the time required for production, allowing the ore to be caved with plenty of time for it to drain and aiding the drying by blowing large quantities of air through the workings.

The rock temperatures to a depth of 850 ft., the present lowest limit of operations, have no material effect on the ventilation requirements. Observations indicate a rise of 1° F. for each 66 ft. of depth in the mine. The rock temperature on the present main operating level of the mine (620-ft. level) is 70° F., so that down to this point the rock temperature would have little or no effect on the mean temperature of the air forced into the mine, as the mean annual temperature is 63.7° F.

The only gases, other than those generated by explosives, are those originating from the timber mat above the caved area, and the pressure system of ventilation readily controls them. These gases are seldom detected, even when the decayed timber is visible on the undercut level and it is not unlikely that they are forced out through the broken capping.

Shafts and Adit

The main operating shaft is of concrete construction, 936 ft. in depth. It is divided into four compartments consisting of two skip compartments 5½ by 6⅙ ft., one cage compartment 6½ by 13 ft., and a ladder, pipe, and counterweight compartment 3 by 13 ft. These compartments are separated from one another by reinforced-concrete walls 8 in. thick. This shaft is well outside of the zone to be caved by mining operations of the known orebody and is convenient to the concentrating plant.

The emergency outlet and main air-supply shaft is concreted throughout, and is composed of two compartments, a smooth-lined airway 4 by 8½ ft. and a manway 3⅙ by 4 ft. equipped with steel ladders and landings; the two compartments are separated by an 8-in. reinforced-concrete wall. The size of the airway was determined by underground ventilation requirements. This shaft is 470 ft. deep and is outside of

the caving zone and on the opposite side of the orebody from the main operating shaft.

Supplies are delivered to the mine through an adit, driven on the railway yard elevation, connecting with the shaft 235 ft. below the collar. This adit is of the same section as the main haulage drifts.

Haulage Levels

The underground development preparatory to the undercut caving system of mining is shown in Figs. 1–4. Connections from the main oper-

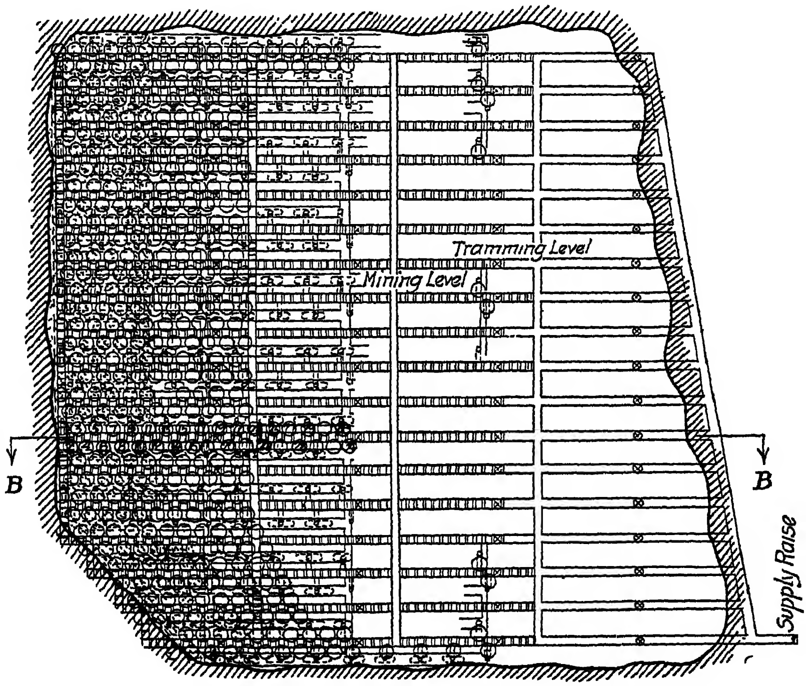

FIG. 1.—PLAN C-C OF FIG. 2.

ating shaft to the orebody are ordinarily driven at vertical intervals of 150 ft. and are termed haulage levels; each haulage level serves two undercut and caving operations or lifts. Haulage drifts are driven parallel to one another, 150 ft. apart, and also parallel to the direction of drawing of the upper lift to be served by them. These drifts are shown in Figs. 2 and 3. The vertical distance between haulage levels was determined largely by the shape of the orebody, which determined the amount of development in waste, both drifting and raising, that would be necessary to serve the ore above. Other factors influencing this spacing were the estimated ton cost and life of haulage levels at various intervals determined by the tonnages to be extracted through them

and the cost of transferring ore through raises and hoisting from various depths.

The spacing of haulage-level drifts 150 ft. apart is fixed by the distance from the haulage level to the upper transfer level, which is 100 ft. and the minimum angle of 53° at which the ore will flow through the inclined raise.

The total haulage-level drifting, including connections from the operating shaft to the orebody, is 1 ft. of drift for each 473 tons of ore served by it. Practically all haulage drifts are timbered with standard drift sets, 10 by 10-in. posts battered 1½ in. per ft., one 9-ft. post and one 9½-ft. post on the ditch side of drift, 10 by 10 in. by 8-ft. cap not framed, but with a 2 by 10-in. spreader 6⅓ ft. long, spiked on the under side.

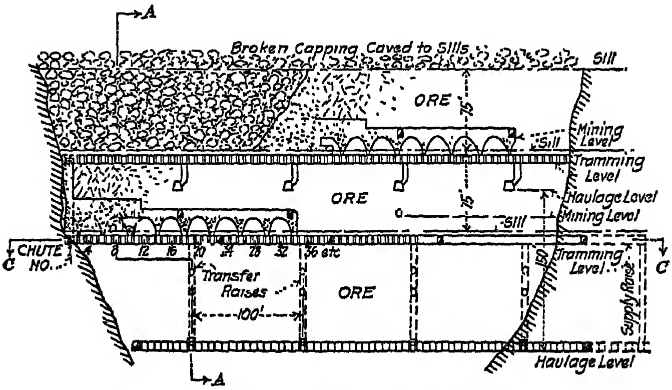

Fig. 2.—Section through B-B of Fig. 1.

Sets are spaced 6¼ ft. center to center, close lagged on top with 2 in. sawed lagging and side lagged where necessary with 2 by 8-in. sawed lagging spaced with 2-in. openings between them.

Transfer Raises

There are two sets of transfer raises from the haulage level: First, the inclined branch raises, as shown in Figs. 2 and 3, are driven to the upper transfer level at intervals of 100 ft. In the harder ground, which will stand without timber, these raises are driven approximately 5 by 5 ft. in section, with stulls and plank to partition off a manway for safety in driving; these stulls and plank are removed after the raise is completed. In softer ground, the raises are timbered with four-post framed sets of 6 by 6-in. and 6 by 8-in. timbers, spaced at 4 or 5-ft. centers, depending on the nature of the ground. The raise is then lined inside of the sets with 2-in. plank, making the section of the raise inside of lining 4 by 4 ft. Under conditions where harder rock is to be transferred through the raise a double layer of 2-in. plank is placed on the bottom. The 10-ft. vertical section immediately below the transfer level is cribbed tightly

with 6 by 8-in. or 8 by 8-in. cribbing. Shallow pockets, 1 ft. deep by 10 ft. long and the full width of the raise, are cut on the lower side of the main raise opposite the junction with the short branches. These pockets are allowed to remain full of broken ore to take up the wear caused by ore entering the main raise from the branches. The minimum angle practicable for these raises is 53°.

The second set of raises are short vertical raises put up later from the haulage drifts, a distance of 25 ft., to serve the lower transfer level; these are shown in Fig. 2. These raises are cribbed 5 by 5-ft. inside measurement. In the first set of inclined raises there is 1 ft. of raise for each 297 tons of ore in place and for the second set 1 ft. for each 1125 tons.

Tramming Levels

The two tramming levels served by a common haulage level are identical, except that the drifts of the first, or upper, level are parallel

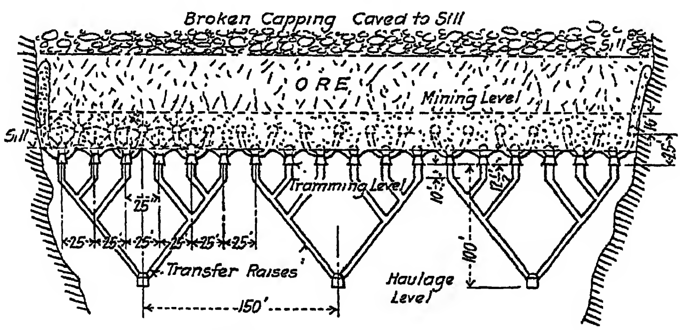

FIG. 3.—SECTION THROUGH A-A OF FIG. 2.

with the haulage drifts, while those of the second, or lower, level are at right angles to the haulage drifts. This is not only necessary to avoid a prohibitive amount of development on the haulage level for the transportation of the ore from the lower lift, but it probably assists in recovering the ore left between the tramming drifts of the upper transfer level and would naturally tend to counteract and break up vertical sections of uneven drawing resulting from the caving of the upper lift. Tramming-level drifts are spaced 25 ft. apart and are connected by small cross drifts at intervals of 100 ft. The crosscuts are a source of trouble when the drawing operations reach them, but are necessary to furnish convenient access to the working places for men and supplies. The crosscuts are driven about 4 by 6 ft. in section and usually stand with little or no timbering until the drawing operations approach within 25 ft. of them, when they are completely filled or bulkheaded with waste timbers. To facilitate the moving of equipment from one crosscut to another as the drawing operations advance, the trackage in the crosscuts is stand-

ardized and made up of turnsheets and sectionalized track, riveted to steel ties.

The tramming-level drifts are first driven small, approximately 4 by 6 ft. in section; ordinarily they will stand at this size indefinitely with little or no timbering; they are later enlarged and drift and chute timber placed in them as they are required for drawing. The practice is to have them timbered up 25 ft. in advance of drawing the operations. The timbering of the tramming drifts consists of 12 by 12 in. sets spaced at 25 in. and 50 in. centers alternately; this makes a chute opening on each side 38 in. wide, spaced at 6¼ ft. intervals. The set consists of two 12 by 12-in. posts 8 ft. long battered 1½ in. per ft; one 12 by 12 in. by 6-ft. cap with 4-ft. spreader spiked on bottom; one 12 by 12 by 55-in. center brace placed between the posts 61 in. above the rail, 6 by 8-in.

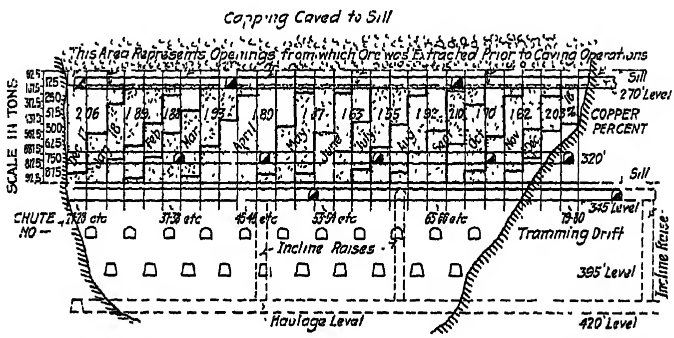

FIG. 4.—SECTION FROM ORE-DRAWING RECORDS, DECEMBER, 1917 TO DECEMBER, 1918.

collar braces, 2 by 8 by 72 in. side lagging, and 6-ft., round, peeled poles 3½ to 5 in. in diameter for back lagging. Chutes are of simple design, built of 2-in. splank and controlled with stopper boards; they are placed in each 38 in. opening of the timbers on alternate sides of the drift; with the drifts spaced 25 ft. apart and the chutes spaced at intervals of 12½ ft. on each side of the drifts, there is a chute for each 156¼ sq. ft. of area.

The tramming level is equipped with 18-in. gage track of 16-lb. rail placed on level grade. Haulage tracks are of 45-lb. rail, 24-in. gage, minimum radius of curvature 23 ft. and a grade of 0.5 per cent. in favor of the load. Drainage water is conveyed on levels through open ditches at one side of the track and is carried between levels in pipes varying from 1 to 4 in. in diameter. Water for machine drilling and drinking is carried into the mine through a 4-in. main, which discharges into 2-in., 1½-in., and 1-in. laterals with final distribution to all sections of the mine through ½-in. pipes, all galvanized. Water pressure is controlled by supply tanks, operating with float valves, so placed as to give a pressure varying from 60 to 150 lb. per sq. in. Compressed air

is conveyed into the mine through a 14-in. main at 90 lb. per sq. in. pressure, and is distributed through 10-in., 8-in., 6-in., 4-in., 2-in., and 1-in. pipes. Ventilating pipe is used in development work only; the sizes used are 12-in. and 19-in. soldered zinc-coated iron pipe, flanged and slip-joint connections made up in 7½- and 15-ft. lengths.

The cost of driving and equipping shafts, stations, haulage level, transfer raises, tramming level, and such drifts and raises, outside of the territory to be caved, which may be required for traveling ways, introduction of supplies, and ventilation, are charged to development and equipment accounts.

Undercutting

Originally the undercutting was identical in all sections of the mine and may be more readily explained by reference to Figs. 1 to 4. First, drifts are driven parallel to the tramming-level drifts on a level 25 ft. above the tramming level. They are spaced 25 ft. apart, but are 12½ ft. to either side of the tramming-level drifts. These drifts are shown in cross section in Fig. 3 by dotted lines above the tramming drift and in longitudinal section in Fig. 2. The method of driving them is as follows: Raises spaced at 25-ft. intervals along the drift are put up on an incline, the tops are widened out to 14 ft. and then the tops of the raises are connected to form the drifts. The mining-level drifts, in this stage, are carried along as far in advance of drawing operations as the tramming-level development will permit. The mining-level drifts are then plugged out in 25-ft. sections to full 8 by 8 ft. area, drilled with fan-shape rounds consisting of from nine to thirteen holes 8 ft. deep, spaced from 2 to 3 ft. apart, depending on the hardness of the ore, and blasted in. This operation is continued until the area to be drawn has been blasted down 50 ft. or more in advance of drawing operations and the undercutting on the mining level is maintained in this relative position. After blasting down, and in advance of the drawing, enough of the broken ore is drawn through the mining-level raises to give the solid back above the blasted mining level an opportunity to break and start caving. This stage of the undercutting is shown in Fig. 2.

In some instances, narrow shrinkage stopes are carried up 25 ft. or more on the boundaries of the area to be caved, for the purpose of limiting the area caved and, in hard ground, to facilitate caving and prevent arching; these stopes are shown in Figs. 2 and 3. The final stage in the undercutting is opening a drift 8 by 8 ft. in section immediately over the tramming drift timbers. Access to these drifts is through the drawing chutes, which have been placed at 6¼ ft. intervals on either side of the tramming-level drift. These drifts are then drilled with rounds almost identical with the rounds used in drilling mining-level drifts and blasted in 25-ft. sections.

Drawing immediately follows the undercutting operations, except that the last two chutes in the undercut area are kept sealed for the protection of the men working in the following undercutting section. It has been found that the first stage of undercutting, which consists of opening up and blasting the mining level, is not necessary in the softer, weaker ore and is now used only in comparatively hard and strong ore.

Drawing of Caved Ore

The actual operation of drawing, which consists of drawing the caved ore through the tramming-level chutes into cars and tramming a maximum distance of 50 ft. to the transfer raises, is so closely associated with the maintenance of the tramming level that these two operations must be considered together.

The area of any drawing section is limited; the minimum is determined by the physical character of the ore and the resultant action of the caving forces or weight of overburden on that ore. Results prove that the softest and weakest ore may be successfully caved in lifts of 75 ft., with a caved area 100 ft. in width and 150 ft. in length; while the hardest and strongest ore caved has required a width of 150 ft. and a length of 250 ft. for a 75-ft. lift and without a doubt better results would have been obtained had the width been 200 ft. or more. The length of the caved area is the dimension parallel to the tramming-level drifts or the direction of retreat; the width of the area is the dimension at right angles to the length.

The maximum area of any drawing section is limited by the cost of tramming-level maintenance and its interference with ore production. To exert a proper control over the drawing of the caved ore—that is, to have the contact between ore and capping approach a plane—it is necessary to maintain an even drawing face across the entire drawing area. When caving areas of excessive width in soft ore, it was more difficult to keep the central tramming lines open and the rate of extraction in these lines fell behind that of the side lines so that it would occasionaly be necessary to abandon clean ore in the central lines in order to even up the drawing face. To avoid this condition as much as possible, a maximum width of 150 ft. for soft ground has been adopted, and in the harder ground widths up to 250 ft. have worked satisfactorily. The length of drawing area is determined by the rate of drawing, which in turn is governed by the rapidity of the caving or breaking up action of the overburden, the daily tonnage required, and the amount of work required to maintain the tramming level. This dimension varies from 100 to 250 ft. for a 75-ft. lift.

With the drawing areas limited, the tramming levels are usually divided into panels 150 ft. in width, extending the length of the area to be drawn. Alternate panels are undercut and drawn. After the first

panels have advanced 200 ft. or more the remaining panels are opened up as required, but are maintained in the same relative position. This allows the capping drawn down by the panels in advance to settle and form side supports for the following panels and protect these panels from excessive weight.

The primary object in the adoption of this method of drawing was to avoid, as far as possible, dilution of ore with barren capping. It was apparent that the drawing of the caved ore from closely spaced chutes into small cars and tramming by hand 50 ft. to transfer raises, and the maintenance of the tramming level and equipment would be more expensive than the drawing of the caved ore directly into transfer raises through a system of branch raises; but as far as available figures show, this additional expense has been warranted by the extraction results obtained.

Supervision and Schedule of Drawing Caved Ore

The drawing of the caved ore is under the direct supervision of an organization that is entirely independent of, but works in conjunction with, the underground operating force. The engineer in charge, with the help of one or two assistants, inspects each chute each day and issues a drawing schedule. This gives the maximum number of cars of ore to be drawn from each individual chute during the following 24 hr. and is based on the amount of capping appearing in the chute, the state of repair of the tramming drift, the total tonnage required, the total number of chutes available for drawing, and the total tonnage previously drawn from each individual chute.

This schedule is delivered to the chute checkers, of whom there are an average of one for eight trammers. The chute checkers instruct the trammers as to the number of cars of ore to be drawn from each chute and check as far as possible the number of cars drawn; at the end of the shift, they report the number of cars drawn and these results are tabulated daily and plotted on drawing sections at the end of each 10-day period. The engineers thus have a daily record of the tonnage drawn and the calculated tonnage remaining in each chute.

Chute checkers may stop the drawing of any chute when the dilution of waste capping becomes excessive, until the engineer has an opportunity to sample or inspect it. The drawing of chutes is stopped when the copper content falls to about 1.25 per cent. This point can be closely approximated by ocular inspection by an experienced man, but is frequently checked by sampling. There is a natural tendency on the part of the trammers to draw chutes that run easily or to draw chutes that are close to or directly above the transfer raises, reducing the length of tram or, in the last instance, doing away entirely with the tramming.

The effect of excessive drawing of chutes is to draw the capping down to that chute; and as the capping runs much more easily than the ore,

it will break into adjacent chutes and leave bunches of clean ore above and capping below. The fact that the trammer after filling his car from a chute must close the chute and stop the flow of ore while he trams his load and dumps it into the transfer raise, is an important factor for ordinarily it is just as convenient for the trammer to draw the next car from the correct chute as from any other. Chutes directly over transfer raises are watched carefully and are sealed until drawing of adjacent chutes is practically completed or are drawn only in the presence of the chute checker. Obviously, any steps toward increasing the tonnage per trammer must be taken with care.

Transfer raises are covered with grizzlies; the rails are 8 in. apart. All the ore must pass through them; little spalling and practically no block-holing is required on the grizzlies. The opening in the drawing-off chute is 21 by 38 in. Chute blasters blast and remove from upper workings boulders and old timbers that will not pass through the chutes; an average of one chute blaster is employed for two trammers.

Rate of Drawing

The rate of drawing is governed by the rate of subsidence of the undercut ore. If drawn too rapidly, capping appears at the chute prematurely. If too much capping has not been drawn down, drawing from that particular chute may be discontinued for a few days, in which time the ore will cut off the channel through which the capping has come down and clean ore can be drawn again. This is especially true in the softer ore, which breaks very fine and if allowed to stand packs until small drifts and raises may be driven in it practically without timber. In the harder ore it is more difficult, and often impossible, to get clean ore once the capping is pulled underneath it.

The maximum rate of drawing after caving is under way has been definitely fixed at 90 tons per chute per day. This would indicate a subsidence of 7.2 ft. per day, as a single chute, tapping a block of ore 12½ by 12½ ft. in section is expected to yield 12½ tons per foot of height of section. The average rate of subsidence is, in the better caving ore, approximately 0.8 ft. per day. This figure is obtained by dividing the height of back caved, in feet, by the number of days elapsed from start to completion of drawing, which varies from 52 to 103 days in different sections of the mine. This period includes the time when drawing was suspended on account of tramming-drift repairs, which in some cases is one-half of the total time.

The chutes are originally installed on alternate sides of the tramming drifts at 8¼ ft. centers and are maintained in this position until the chute is finally sealed. They are then moved across the drift and the small pillar of ore remaining between the original chutes on that side of the

drift is drawn. The chutes in the changed position yield from 15 to 20 per cent. of the total ore expected from the chute.

Extraction Records

The records obtained by daily inspection from the completed drawing of 221,800 tons of ore through 271 chutes show that 83 per cent. of the original expected tonnage was drawn clean, that is, withoutad mixture of capping; 14 per cent. additional was drawn with less than 10 per cent. dilution, and the balance drawn varied from 10 to 50 per cent. waste.

The extraction results obtained from completed sections of the mine based on tonnage and assay values, calculated as previously described, are as follows:

1. An orebody, the maximum dimensions of which were 500 by 500 by 325 ft. high. The ore was disseminated chalcocite in granite-porphyry, a soft weak ore especially adapted to caving. This orebody was mined in three lifts, varying from 50 to over 200 ft. The expectancy was 3,772,254 tons of ore assaying 1.959 per cent. copper, the extraction was 3,954,320 tons of ore assaying 1.763 per cent. copper. The tonnage extraction was 104.8 per cent.; the grade extraction 90.0 per cent.; and the copper extraction 94.3 per cent.

2. The first lift in the main orebody is the typical sulfide ore in schist. This orebody had been mined by top-slicing to an even horizon and undercutting was begun 75 ft. below the level at which slicing was discontinued. It was supposed that the heavy timber mat formed by top slicing would aid the caving operations, especially in preventing the admixture of the capping. The results obtained from caving the orebody and from caving portions of the main orebody outside of the sliced area, however, indicate that the timber mat not only fails to assist in separating the ore and capping but is a disadvantage in drawing the caved ore. The timber mat reaches the drawing chutes before all the ore has been extracted and large quantities of it must be blasted out of the chutes and transported to the surface.

The expectancy in the first lift, which has been completed, was 2,464,303 tons of ore assaying 2.19 per cent. copper; the extraction was 2,483,581 tons of ore assaying 1.811 per cent. copper. The tonnage extraction was 100.78 per cent.; the grade extraction, 82.70 per cent., the copper extraction 83.34 per cent.

In drawing this lift it was at times advisable to abandon fairly large quantities of clean ore because of excessive maintenance costs on tramming and haulage levels and to maintain an even drawing face. At the time, it was thought that this abandoned ore would be recovered on the second lift. To date, drawing of 62 per cent. of the next lower lift has been completed. Extraction results based on the expectancy of the ore in place between the two undercuts plus the ore abandoned on the first

lift indicate that all the abandoned ore will be recovered on this lift as anticipated, and the final results will be as good, if not better, than those obtained from the other orebody cited.

The extraction results are obtained as follows: The tons of ore drawn from individual chutes are obtained by allowing a fixed amount of ore for each car drawn. This is corrected at the end of each 10-day period, by applying to these estimated weights a factor that will make the total agree with the total production of the mine as determined by the automatic weighing devices at the concentrating plant. The grade of the ore drawn is determined in the following manner: Hand-grab samples are taken from each haulage car as it is loaded from the transfer raise. These samples are assayed daily and the average is corrected at the end of each 10-day period by applying a correction factor, which will make the average of all ore drawn for the period coincide with the grade, as determined by automatic sampling at the mill. The results obtained from these grab samples will average close to 8 per cent. above the final result obtained by automatic sampling at the concentrator, and they serve, to a certain extent, as a guide in drawing.

The grade of the product from any one section obtained in this manner is subject to question, but the total product is accurate. For this reason the results obtained on the completed parts of the second lift of the main orebody are subject to question and have not been included in this paper.

Equipment

Drilling Machines

Three types of drilling machines are in use: Heavy mounted drills with independent rotation of the "Turbro" type using 1¼ in., round, hollow, drill steel; air-feed hammer drills of the hand-rotated type, using ⅞-in., quarter-octagon, solid and hollow drill steel; light unmounted self-rotating hammer drills of the jackhammer type, using ⅞-in., hexagon, hollow steel. All bits are straight-faced crossbits, double-tapered 5 and 14°, with a change of gage of ⅛ in. for following lengths.

Explosives

Two varieties of explosives are in use: 30 per cent. standard dynamite and 30 per cent. gelatin, both detonated by means of 6X blasting caps. All firing is by means of standard double-taped fuse, except in shaft sinking when blasting with delay-action electric detonators fired by battery is employed.

Timber

All timber is framed or cut to lengths required on the surface, and is loaded on trucks, which are taken to the shaft through the supply adit.

Storage spaces for more than 24-hr. requirements are provided on each haulage level. Timber for the upper tramming level is routed through the haulage level 50 ft. above it, whence it is delivered through a timber slide inclined at an angle of 28° from the horizontal. Timber for the lower tramming level is hoisted through supply raises, spaced approximately 300 ft. apart from the haulage level 25 ft. below it. The length of cage permits the transportation of all timber and other supplies, up to 12 ft. in length, from the surface to any part of the various haulage levels in the same truck without re-handling.

Mine Cars and Locomotives

End-dump cars equipped with roller bearings are used in the hand-tramming levels; they are of two sizes, one with a narrow box of 16 cu. ft. capacity for use in small openings and one with a box of 20 cu. ft. capacity; the trucks of both are identical. The haulage-level cars are gable bottom, equipped with roller bearings, truck and drawbar springs, and are of 60 cu. ft. capacity. Six-ton, single-motor trolley locomotives operating on 220-volt d.c. are used for ore haulage. Four-ton storage-battery locomotives are used for hauling supplies and, to a limited extent, in haulage-level development work.

Ore Pockets

The ore trains pass over and dump directly into a 700-ton ore pocket adjacent to the skip compartments of the shaft. They are dumped by hand-operated levers controlling the car doors. The pocket is built of concrete, lined with steel rails. The ore is drawn from the bottom of the pocket through two measuring hoppers into the two skips. The flow of the ore through each hopper is controlled by two air-operated gates.

Man Hoist

All men, supplies, and waste rock are transported through the shaft in a single-deck cage 5¼ by 12½ ft. inside measurement. The cage is operated with a counterweight by a single-drum hoist, driven through a single-gear reduction by a 250-hp. 440-volt, 25-cycle, three-phase, electric, hoist motor, equipped with liquid controller.

The hoist is equipped with slow-down, overspeed, and overwind safety devices; in addition, electric limit switches are installed on both cage and skip roads on the head frame.

Hoisting Ore

The ore is hoisted in two skips, each of 10-ton capacity operating in balance. The skips are operated by a double-drum geared hoist with a

capacity of 12,000 tons hoisted 796 ft. per 24 hr. The hoist is driven by a 1400-hp. d.c., motor through a single reduction of herringbone gears. The current for the hoist motor is supplied by a motor-generator set consisting of a 1000-kw., d.c., generator direct-connected to a flywheel and an 1100-hp. variable-speed induction motor operating on a 6600-volt, 25-cycle, three-phase current. Brakes and clutch are operated by hydraulic thrust cylinders and the control of the hoist is through a slip regulator of liquid type operated by a regulating motor. The hoist is equipped with overwind and slow-down safety devices.

Pumps

The mine water is pumped to the collar of the main shaft from a central station 850 ft. below the surface by two 5 by 12-in. quintuplex electric driven plunger pumps, each having a capacity of 150 gal. per min.

Air Compression

Compressed air is furnished at a pressure of 90 lb. per sq. in. by two steam-driven compressors each with a rated capacity of 4000 cu. ft. of air per minute.

Ventilation

In any mine in which there is almost constant blasting throughout the shift ventilation becomes an important factor in the operations. The requirements for the proper ventilation of all workings were carefully considered when studying the plan of development but a complete description of the ventilation system and equipment in use could not well be given here because of its length. Artificial ventilation is used. Air is forced into the mine by two reversible multivane blowers, which deliver 180,000 cu. ft. of air at about 4¼-in. water-gage pressure through two air shafts. The course of the air is through the tramming-level drifts toward their abandoned ends, thence down to the haulage level through abandoned transfer raises, thence exhausting to the surface through two upcast shafts, one of which is the main operating shaft. This is contrary to general practice. The other shaft is equipped with an exhaust fan, which may be started up at short notice. Several types of doors for control of ventilation are in use. Small multivane fans of 6000-cu. ft. capacity direct-connected to electric motors are used for ventilating exploratory and development workings in remote sections.

Lighting

The shaft stations, haulage drifts, tool rooms, and manways are lighted by electricity. The current is that used for the trolley lines, but

the important lights are on a separate circuit. The emergency outlet is lighted by an entirely different circuit. The use of carbide lamps by underground employees is compulsory; lamps are sold to them at cost.

Signals

Signals from shaft stations and loading pockets are transmitted by electric pull bells. There are hand-bell lines in cage and skip compartments for use in shaft inspections and emergency. A telephone system is installed with instruments at all stations and numerous other points throughout the mine.

Records of Unit Production

The figures given are based on 8-hr. shifts. To compare these figures with records based on work per hour they should be divided by 8, the number of hours in a shift, or by 6¾, the average number of hours actually worked per shift, allowance being made for traveling to and from work and for lunch. These records are the result of operations from Jan. 1, to Nov. 30, 1922, inclusive, during which time the plant was operated at 82 per cent. of its rated capacity.

The total production was 2,006,840 short tons, dry weight.

All work was performed by day labor, excepting development drifting and raising, which was part contract, part day labor with bonus, and part straight day labor.

Tons per man per shift for all men in stoping, 10.84. (Included in this figure is all labor used in undercutting and transferring ore on tramming level and maintenance of tramming level and transfer raises, including direct supervision.)

Tons per man per shift for all men on development in ore, 2.73. (This includes breaking, timbering, and delivering broken rock to the nearest tramming level.)

Tons per man per shift for all men on development in waste rock, 1.52. (This includes the hoisting and transfer of the broken rock to the main operating shaft.)

Tons of ore per man per shift for all underground labor, 7.58.

Tons of ore per man per shift for all surface labor in mine department except office force, 92.1.

Tons of ore per man per shift for mine organization, 6.82. (Included in this figure is all labor directly chargeable to the mining department, but not proportionate charges of the general management, general office, power plant, mechanical, and supply departments.) The mining operations end with the delivery of the ore into the supply bin of the coarse crushing plant.

The classification of labor in percentages follows:

Underground	Per Cent.	Surface	Per Cent.
Machinemen	23.72	Blacksmiths and drill sharpeners	16.95
Muckers	8 68	Machinists and repairmen	16.90
Trammers	10.65	Timber framers and saw filers	7.17
Chute blasters and checkers	9.66	Hoist enginemen and oilers	3.60
Timbermen and helpers	30.00	Top landers	3.32
Motormen and trainmen	7.34	Janitors	6.09
Cagers and skiptenders	1.44	Laborers	14.48
Nippers	2.56	Miscellaneous labor	2.13
Trackmen, pipemen, etc	2.42	Divided (part under ground and part surface)	19 36
Bosses	3.53		
	100.00		100.00
Or, 89.88 per cent. of total labor		Or, 7.41 per cent. of total labor	

Office

Including mine superintendent and assistants, general foremen, master mechanic, mining engineers, clerks and timekeepers.................................. 2.71 per cent. of total labor

The labor turnover for the period was 15.1 per cent.; this equals the total terminations minus the unavoidable terminations (deaths) divided by the average number working daily.

Records of Units of Supplies Used

Gelatin and ammonia dynamites, pounds per ton of ore	0.33
Sawed timber, board feet per ton of ore mined	3.59
Pole lagging and small stulls, linear feet per ton of ore mined	0.33
Electric power, kilowatt-hour per ton of ore.	
Haulage	0 270
Hoisting	1.333
Pumping	0.082
Ventilation	0.672
Lighting	0.067
Shops and miscellaneous	0.174
Total	2.599
Cubic feet of compressed air per ton of ore	988

The cost of all supplies used other than those just mentioned amounts to 15.7 cents per ton of ore.

Safety and Welfare

The regular training in first aid and mine rescue is under the direction of a district association. In addition to the compulsory training of bosses, regular training of employees is given to all who are willing and interest in the work is stimulated by the payment of a cash bonus monthly

to each member of organized teams in both first aid and mine rescue. The men qualify for membership in the first-aid teams by competitions. The members of the rescue teams are selected from among those completing the voluntary training course.

All safety work is incorporated in the duties of the operating force. Safety inspectors are employed for specific duties.

The management keeps in touch with and renders aid to injured and sick employees through the employment department. Vegetable-garden plats with expert supervision are furnished employees. The company has constructed and, to a large extent, maintains two branches of the Y. M. C. A., a staff club house, and a staff mess house. A department store is operated under the direction of the management on a profit-sharing plan, in which all profits from the business are returned to those employees who have been in the service of the company continuously during the last 3 months of each period of 6 months, in the proportion of their purchases. The administration of some of these activities is participated in by the employees through an employee's committee, which is composed of representatives from the various departments of the plant, elected annually. Modern dwellings at the mine are maintained for and rented to the members of the operating staff and to a small proportion of the other employees.

Estimation of Ore Reserves and Mining Methods in Alaska Juneau Mine

By P. R. Bradley, Juneau, Alaska

(New York Meeting, February, 1924)

The property of the Alaska Juneau Gold Mining Co., covering the Juneau gold belt for over a mile in length, consists of 136 claims and 24 mill sites; these are near Juneau, Alaska, in what is known as the Harris mining district.

Gold was discovered in the district in 1880, by Richard T. Harris and Joseph Juneau, who had been prompted to explore the section by rumors they had heard among the Indians. Gold was first found on the beach at the mouth of Gold Creek and further search revealed the lode deposits in the Silver Bow basin, about two miles inland. The lode locations made immediately thereafter constitute the principal area of the Alaska Juneau group of claims. Gradually news of the strike spread and, during the late eighties, Silver Bow basin became a substantial producer of gold, both from placer work and from quartz milling. Numerous small mills of various types were erected to crush high-grade float and stringers gouged to shallow depths.

Placer mining gradually ceased and the pioneer milling operations gave way to larger and better mills, which in time were the predecessors of the two large mills built by the Alaska Gastineau Co. and the Alaska Juneau Co. on the shores of Gastineau Channel. This evolution from arrastres to small mechanical mills, thence to the two large-capacity mills on Gastineau Channel, although a natural sequence of events, had a most important bearing on the development of the district. Knowledge of the character, extent, and value of the ore deposits gained by these early operations could not have been acquired otherwise.

Spencer shows[1] that the early mills were: Arrastres in 1881–82–83–84–85–86–88; a 5-stamp mill in 1882; a "newly devised grinding mill" in 1884; two Huntington mills in 1886; a 10-stamp mill in 1889; a revolving Dodge mill in 1890; a 5-stamp mill in 1891; and a 20-stamp mill in 1893.

He also estimates the gold production to the end of 1893 as $1,000,000 from lode mining and $1,250,000 from placer mining. No estimate can

[1] U. S. Geol. Survey *Bull.* 287.

be made of the amount and grade of the ore milled, but no doubt the grade was high, as only the best surface quartz was gouged out and milled. An authentic record states that $1200 in gold was recovered from a float quartz boulder 3 ft. in diameter.

In 1896, the owners of the Alaska Juneau property, having been encouraged by results obtained from their own 5-stamp mill, as well as by results of neighboring operations, built a 30-stamp mill. In 1898, a 15-stamp mill was built on the Ebner property. In 1907–08, a 100-stamp mill was built by the Perseverance Co. The amount and grade of the ore treated by these three mills was:

Mill	Tons	Average Assay Value
Alaska Juneau	407,254	$1.71
Ebner	75,000	3.50
Perseverance	300.000	2.05

The ore for the Alaska Juneau and Ebner mills was mined from open pits; that for the Perseverance mill came from underground stopes. All the ore was from selected areas but results indicated that a tremendous tonnage of low-grade ore was available.

During the years that this ore was being mined and milled a close study was made of all local conditions and problems, and valuable economic and technical data were gathered. The conclusions obtained therefrom may be summarized as follows:

1. That the Juneau gold belt is an integral portion of the coast range geology, and is of great length, of variable width and of irregular and erratic mineralization.

2. That the belt itself is slate and similar sedimentaries with intrusions of metagabbro.

3. That the belt is divided into bands of poor definition.

4. That in the vicinity of Juneau the various bands converge to a limited width, become better defined, and constitute a broad lode.

5. That the gold in the belt is due to the presence of quartz.

6. That the broad lode in the Juneau vicinity contains an enormous tonnage of low-grade ore.

7. That this grade of ore could be mined and milled at a commercial profit, when operating on a large-scale all-year basis.

8. That large-scale all-year operations demanded low-cost underground mining, sea-level mills, and a continuous supply of cheap power.

Climatic and other conditions rendered all-year operations impossible at the high altitudes where the various early mills were situated. Therefore the proposal to change from seasonal water-power operations to all-year operation involved low adit-level outlets from the mines to suitable mill sites on the beach, these adits serving at the same time to tap the ore at depths as much as 2000 ft. below the surface.

The first deep-level adit on the Alaska Juneau was planned in 1898 and started in 1899, but it was not definitely arranged for until 1910. In that year, F. W. Bradley made a contract to drive a tunnel 420 ft. above sea level for a distance of 6500 ft. to a point beneath the surface workings, and then to raise 800 ft. to the surface. This work was started in 1911 and was completed in about two years. In 1913–14, the Alaska Juneau Co. built a 50-stamp pilot mill on the hillside near Juneau, and in 1916 a new mill was built, with a rated capacity of 8000 tons per day; but, owing to the failure of experimental features (for which the present management is not responsible), its actual capacity proved to be less than 4000 tons per day. The milling costs proved about four times higher than the normal milling costs in the district.

The Alaska Gold Mines Co. was organized in 1912 and in that year began a 10,000-ft. adit at a point about 670 ft. above sea level. In 1914, this company built a mill at Thane; the rated capacity was 6000 tons per day, but its actual capacity proved to be much greater.

In 1912, the U. S. Smelting, Refining and Mining Co. acquired the claims owned by the Ebner Co. and drove a 4000-ft. adit from a point about 350 ft. above sea level; this adit was connected to the surface, and considerable drifting and crosscutting was done. No mill has been built on this property.

Geology of the District

The Juneau gold belt lies between the diorite core of the coast range mountains and the greenstone and chloritic schists on the northeasterly shore of the Gastineau Channel. The rocks occupying this interval consist of sediments and their various derivatives, together with irregular intrusions of metagabbro. The sediments have been folded, intensively plicated, and subjected to the compression that developed the slaty cleavage. This cleavage is generally coincident with the strike but not with the dip. The rocks making up the sedimentary series consist of phyllite gray slates, graphitic slates, laminated quartzites and sericitic schists. The metagabbro intrusions are fairly abundant in the slates, and extend to a lesser degree into the schists and greenstones; they occur in the form of sills, small laccoliths and dykes. The gabbro, in horizontal section, forms a pattern of roughly parallel bands, of variable width and generally lenticular.

Gold occurs chiefly, if not wholly, in quartz stringers and gash veins in the slate and metagabbro. The quartz is irregular in form and disposition, following the strike of the slate cleavage in a general way, but usually more vertical than the dip. Clean quartz will average $6 per ton within the areas of commercial ore; without these areas there is an abundance of quartz carrying little or no gold. The gold itself is erratically distributed and of wide variation in size. The gold recovered in the

Alaska Juneau mill ranges from nuggets, with a maximum dimension of 0.75 in., to the finest dust.

Associated minerals are galena, sphalerite, pyrrhotite, and pyrite. Galena and zinc are usually highly auriferous; the pyrrhotite will assay about $6 per ton and the pyrite less. Silver is recovered from Alaska Juneau ore in the proportion of 1 oz. to each 1.612 oz. of gold recovered.

Exploration, Sampling and Estimating Methods

The gradual expansion of mining and milling operations on the belt had an important bearing on present-day operations. Each step in the progress was the result of the preceding step, and the final operations of the Alaska Juneau Gold Mining Co. and of the Alaska Gold Mines Co. were based on accumulated information rather than on a systematically developed tonnage of which the assay value had been determined by any of the hand-sampling practices commonly in use. Each of these companies had acquired, in its own way, such data as were deemed necessary for the determination of the amount and value of available ore, but in neither case was the operation based on a predetermination of values arrived at exclusively through hand sampling.

Theoretically, it should be possible by hand sampling to determine the average assay value of any gold ore, but on the Juneau gold belt the gold is unusually coarse for lode gold, its distribution in the quartz is erratic, and the quartz itself is irregularly disposed throughout the orebodies. It is a difficult problem, therefore, to determine in advance, by any system of hand sampling, the average gold content of these ores. Furthermore, the development work done in any one of the mines on the belt, in advance of milling operations, was not sufficient to permit a reliable determination of average assay values by hand-sampling methods exclusively. The local problems met in sampling and estimating values differ in degree only from those met in gold mines elsewhere.

In order to forecast within an allowable degree of error the average assay value of exposed ore by any method of hand sampling it is necessary that: (1) The samples be correctly taken; (2) a sufficient number of samples be taken; (3) the samples be properly manipulated and correctly assayed; (4) the average value as finally determined be for the identical ore that will be mined; (5) all data obtained be correctly interpreted.

Samples Correctly Taken

Samples taken by hand can be either muck samples or moiled channel samples. Muck samples may be helpful to the operator, but to an examining engineer they are of little, if any, value. The examining engineer must resort primarily to channel samples, and the information obtained through these may be supplemented or modified by actual mill returns or by special mill tests.

Moiled channel samples may be cut large or small, sectional or continuous, as the engineer deems best. A correctly cut sample will include all materials within the sampled section; and these materials will be included in the exact proportions in which they are present in the rock mined from the section sampled. Correct cutting of samples is a simple matter; incorrectly cut samples or toleration of them signifies incompetence.

The average gold assay value of the ore milled during the history of the Alaska Juneau mine is 96 cents per ton, but many samples taken from within stoping areas exceed this average by several thousand per cent. Any practical method of hand sampling this ore will produce this great variation of values.

Systematic channel sampling in the Alaska Juneau mine is considered an unnecessary expense, and is not now practiced. The value of ground not already known through actual mining is gained by grab samples taken from the muck during the progress of development work. The assay results of such samples are interpreted in the light of experience and knowledge of the ground. In the Alaska Juneau mine the chief purpose of sampling is to determine the grade of what the miner recognizes to be ore, and to make a record of the information. Grab sampling more truly reflects the personality of the sampler than any other system of hand sampling and it is necessary to select for this work a man of suitable temperament. Experience in the Douglas Island mines proves that, for that group at least, it is possible to take grab samples with satisfactory results. The record from these mines is:

	Treadwell Mine 17 Years	700 Mine 14 Years	Mexican Mine 16 Years	Ready Bullion Mine 15 Years
Average assay muck samples.....	$2.37	$2.20	$2.88	$2.72
Recovery plus tailings...........	2.47	2.33	2.83	2.24

The method of mining in the Douglas Island mines required a great amount of preparatory work, and from this work a large number of samples were obtained throughout the orebodies. In the Alaska Juneau mine the amount of preparatory mining work is relatively small and muck samples are obtained from a small area only. Nevertheless, in recent years, these muck samples have checked very closely the mill returns on that part of the orebodies from which the samples were taken.

The selection of the method to be used in sampling a gold mine is one problem for the management, and another problem for an examining engineer. The management has or should have an "experienced eye," and in most cases this will tell what is and what is not ore; and the management should know what factor to apply to the assays, if any factor is necessary. This is because of the very great advantage of milling ore of which the assay value has already been estimated.

Sufficient Number of Samples Necessary

If a mine were of uniform grade throughout, one sample correctly taken, manipulated, and assayed would be sufficient for the determination of its value, and it follows that the more varied the values and the more irregular their distribution the greater should be the number of samples. Even in such extreme variation and distribution as is found in the Juneau gold belt, there is no reason to believe that the law of averages will not prevail, provided a sufficient number of samples are taken. No mine on the Juneau gold belt has been developed sufficiently to expose the faces required to give the correct number of samples for a predetermination of average value by hand sampling without milling experience; for the capital outlay required for the necessary development work, to determine average value by sampling exclusively, would be prohibitive.

The merit of any system of hand sampling lies not in the fact that each assay correctly stands for the value of the section from which the sample was taken, but in the fact that the calculated average of all the assays of samples taken from a particular block should represent the value of that block. The essence of any system of hand sampling lies in averages; obviously, then, the larger the number of samples the more reliable the result. Just how many samples to take, or what sample interval to adopt, is a matter of judgment; there can be no fixed rule, and each mine is a separate problem.

Before the present practice of taking a large number of channel samples at frequent intervals and relying on averages, it was the custom of some mining engineers to take a few samples blasted down from sections separated by relatively wide intervals, and to use the assays of these samples in estimating value. This practice placed too great a burden on a single sample and a single assay, and consequently it did not survive.

Correct Manipulation and Assaying of Samples

Theoretically, if a sample has been correctly taken, correctly manipulated and correctly assayed, the assay result should exactly represent the value of the section from which the sample was cut; practically, however, such will not be the case, for it is most unlikely that all three operations will be perfect. Samples taken with care, reduced in bulk accurately, and skillfully assayed, will approximate the correct value for the section from which the sample was taken, and errors should be compensating. It has been shown that, from the point of view of straight sampling, many samples are necessary in order to obtain a reliable average; and in view of the chances for error in the two subsequent operations—manipulation of the sample and assaying—the necessity of depending on averages becomes more apparent. In view of the

TABLE 1.—*Results of Sampling Two Crosscut Tunnels in Alaska Juneau Mine*

Crosscut	Length of Section, Feet	Average Assay of Moil Samples	Average Assay of Muck Samples	Mill Returns Free Gold Plus Tails
	60	$0.00	$5.21	$0.54
	25	0.66	0.00	0.99
	83	0.77	1.22	0.67
No. 1.............	63	0.87	0.35	1.06
	72	0.67	1.03	1.22
	40	0.00	0.45	0.69
	55	0.49	1.09	0.56
	173	0.43	2.07	1.60
Average.........		$0.49	$1.43	$0.92
	95	$0.58	$3.51	$0.54
	50	0.31	0.51	0.94
	40	1.12	2.27	1.13
No. 2.............	45	0.07	1.79	0.63
	167	0.60	1.12	0.62
	76	0.73	2.21	1.36
	41	0.06	0.86	0.64
	97	2.13	0.08	0.94
	173	1.48	1.78	1.65
Average.........		$0.79	$1.57	$0.90
Weighted Averages				
No. 1.............	571	$0.499	$1.657	$1.012
No. 2.............	784	0.943	1.577	0.997

TABLE 2.—*Results of Ten Assays on Samples from Alaska Juneau Mill*

Assay No.	Sample No. 1	Sample No. 2	Sample No. 3	Sample No. 4
1	$0.41	$0.83	$0.62	$1.86
2	0.41	0.20	1.45	1.24
3	1.24	1.03	1.65	2.27
4	1.45	1.45	5.37	1.24
5	0.00	0.83	5.37	2.89
6	0.41	0.83	1.24	1.24
7	1.24	0.00	2.07	0.62
8	0.20	0.83	4.96	1.65
9	0.41	0.62	3.93	0.83
10	0.20	0.41	3.93	1.65
Average...........	$0.59	$0.70	$3.34	$1.55

very erratic distribution of gold in the Alaska Juneau ores, and its extreme degree of coarseness, a discussion of the assaying problem is of special interest, because it has a direct bearing on the estimation of values.

The entire bulk of a moiled channel sample cut in the Alaska Juneau mine may contain a sufficient quantity of gold to return an assay of, say, $2 per ton. If this quantity of gold occurs in one piece, as it can and frequently does, then with each reduction of the bulk, before final grinding, the chances are equal that the gold will go with the reject, or stay in the sample. If it goes into the reject the sample will assay nil; if it gets into the cut that is given the final grinding, the result will be an extreme assay. If a check sample were cut four times as large, it should contain four times as much gold, with greater chances for a correct sample after reduction; but, practically, that will not be the case, because, owing to the erratic distribution of the gold, the larger check sample may not contain any gold at all.

If the gold in an original moiled sample occurs as any odd number of equal-weight particles, any cut after quartering will not accurately represent the value of the original sample; likewise, if the contained gold is any even number of equal-weight particles, the reduced bulk cannot be further divided with accuracy before grinding, after it contains an odd number of particles. Differences arising from this fundamental proposition should be compensated somewhat in ordinary gold-ore bulk samples by variation in weight of gold particles. In extremely fine gold, differences would be reduced to a minimum; and, conversely, the coarser the gold, the smaller the chance that the final cut will accurately represent the original sample.

Two of the early crosscut tunnels in the Alaska Juneau mine were sampled in sections by grab samples as the work progressed and later by moiled channel samples. Afterward the sides of the crosscuts were slabbed off and milled. The results were as shown in Table 1.

In the course of some experimental work in the Alaska Juneau mill, a number of samples were given special attention. The final cut of these samples was ground to pass a 100-mesh screen and ten assays were then made of each sample; Table 2 shows the results, each column of figures showing the ten separate assay results from a single sample.

To one not familiar with these ores, the irregularity of results shown in the mine-sampling tests would indicate that the moiled samples were carelessly taken, and that muck samples are not to be relied upon; the wide range in variation shown in the ten separate assays of the same sample would suggest incompetent assaying. The assay office making all the foregoing assays has had more experience in assaying low-grade ore of this character than any assay office anywhere. If, then, it be assumed that the assaying is as skilfully done as it can be by standard practice, and that the variations shown in the series of ten separate assays

of one sample are unavoidable on this class of ore, it becomes a question as to whether the variations between moil and muck samples and between these and mill returns arise out of the assaying or out of the sampling.

The real point is, are variations in assaying this class of ore avoidable? The evidence points to the belief that they are not. If a cube of gold small enough to pass a 100-mesh screen is included in one assay-ton of pulp, it will increase the assay result $1.12 per ton. As long as the gold in a sample exists as free particles of uneven size, it is not reasonable to suppose that it will be so evenly distributed in a quantity of ground pulp that ten single assay-tons having an identical weight of gold can be selected from the pulp; yet assaying practice, and too frequently the judgment of results, is based on the theory that this can be and is done. The assay of a single sample of ore containing coarse gold is at best an approximation of its value; real dependence must be based on averages and not on the results of single assays. This conclusion is verified by a test made in the Treadwell assay office. In that test, 28 samples were halved by the most approved type of automatic sampler. The final pulp was passed through a 100-mesh screen and each sample of the duplicate sets so labeled that its identification by the assayer was impossible. The variation in duplicates ranged from zero to $4.96, the average variation being $1.43, or 41.3 per cent.; yet the average of one set was $3.50 and of the other set $3.41, the difference being a quantity too small to be weighed on the ordinary assay balance.

The accuracy of assaying will increase as the quantity or fineness of the gold increases. Ten duplicate sets of Treadwell concentrates average $68.81 and $68.77 respectively, the average variation being $5.21, or 7.6 per cent.

The conclusion is that the ordinary standard fire assay for gold is a determination of precision only in so far as it has to do with the pulp that goes into the crucible.

Final Determination of Average Value

As finally determined, the average value should be for the identical ore that will be mined. Many causes may intervene to reduce the grade of the ore milled below its estimated value; there are but few things that will increase the estimated grade and these seldom, if ever, happen.

The chief cause of reduction in grade, if this has been correctly ascertained, is unforeseen dilution from inclusions of wall rock, and the inclusion of low-grade vein matter as a result of careless mining. The engineer who defines the boundaries of orebodies does so with a far greater degree of accuracy than does the miner who mines them. Errors in this particular respect are due both to the engineer who values the ground and fails to

take dilution into consideration and to the operator who through carelessness, or the adoption of a wrong mining system, permits the unexpected dilution to occur.

Correct Interpretation of Data

Generally, mill returns plus tailing losses work out less than the estimated value. Any one or all of several causes may be accountable for this result, but the chief cause, no doubt, is failure to interpret data correctly. The real problem in sampling and estimating work lies not in obtaining data but in its interpretation. Faulty use of data may result in underestimating the volume of material that will be mined, or in overvaluing the ore in place, or in a combination of the two errors. Underestimation of volume has been discussed.

Assay returns may represent all errors that have preceded, such as samples taken by an incorrect method, samples incorrectly taken by a right method, careless manipulation of the samples, as well as faulty assaying. Judgment and experience are required for the consideration of individual assay results, and of collective results as well. In the early days of the Camp Bird mine, samples were taken from a newly exposed orebody over a length of 600 ft. The samples showed erratic results, and at one section repeated cuts showed unusually high returns. The acceptance or rejection of the average of these returns meant a difference of $5 per ton over the entire 600 ft. Arbitrary rejection or modification of high assays may be safe, but it is not scientific; assays can be extremely high and extremely low, and be right. If there is any doubt as to the returns of any particular assay, the doubt should be eliminated, not the assay.

In estimating the value of an ore deposit acknowledged to be of low grade, there is the temptation to reject high assays, but low-grade gold deposits are by no means uniform and for each area lower than average grade there must be a corresponding area above average grade.

Conditions Governing Selection of Mining Methods

As previously stated, the Juneau gold belt is divided into ore bands of poor definition. The most easterly workings on the belt, those of the Alaska Gastineau Co., disclosed three separate bands: the foot wall or Ground Hog band, 70 ft. thick; the Perseverance band on which the recent mining was done, also 70 ft. wide; and the No. 2 band, 100 ft. wide. These bands, aggregating 240 ft., are found over a width of 2000 ft. Westerly in the adjoining Alaska Juneau ground, four bands have been found which have a total right angle thickness of 755 ft. These bands are convergent in their westerly trend and terminate on the plane of Silver Bow fault. The four bands have better definition at the fault plane than at the easterly boundary line where they enter the property.

Along the property line, the width of the bands and intervening country is 1600 ft., and at the Silver Bow fault this width is reduced to 1300 ft. The distance from the easterly property line to Silver Bow fault is 2400 ft. The bands are slate or slate with metagabbro intrusions, together with gold-bearing quartz masses, stringers, and gash veins. The bands themselves are not all ore, but all commercial ore is found within the bands. They have no uniform width either in horizontal or vertical section. The ore has no hard and fast boundary except where cut by faults, and profitable mining ceases on a vague and indefinite line where the auriferous stringers and gash veins become too few. The bands are separated by country too lean in quartz to make ore. These lean or waste bands are just as lacking in uniformity as the ore bands.

The throw of Silver Bow fault was downward and westerly, the components of motion being about equal, and the apparent horizontal displacement is about 2000 ft. Being later than all the rocks and later than the ore, the fault has had the effect of dividing the mine into two parts. The occurrence of the ore bands on the easterly side of the fault has already been described. On the westerly side, there apparently has been a coalescence of the bands without the identity of any one band being apparent, unless it is the foot wall or Nugget Gulch band. The belt passes out of the property at the west property line and enters the Ebner property of the United States Smelting, Refining & Mining Co.

For mining purposes, a band can be considered as a definite channel of values, but actually this is not the case; the distribution of values throughout each channel or band is by no means uniform. There is an abundance of evidence to this effect in all the mines on the belt, and the irregularity of values extends both vertically and horizontally. No general law has made itself evident; that is to say, there is no system of repeated orebodies within the bands, no arrangement of horizons of characteristic leanness or richness, and no marker of any kind that will indicate that a certain unknown part of a band will be good, bad or indifferent. Some bands are known to be better than others, and parts of the same band are definitely known to vary greatly in gold content. The gold values may diminish and the amount of quartz remain the same; but, as a general rule, quartz that is gold-bearing contains sulfides of lead, zinc and iron, and conversely the absence of these sulfides denotes an absence of gold in the quartz, and marks it as low grade. Hence an intimate acquaintance will enable the eye to distinguish between payable areas and those not payable.

The selection of a mining method for the mines on the belt resolves itself immediately into consideration of selective mining and wholesale mining.

Selective Mining.—Small-scale selective mining has been tried in a number of instances, and while the result in some cases has been a small

operating profit, none of the enterprises was a commercial success. Large-scale selective mining means a large number of small stopes or a smaller number of large stopes; in either case stope control is necessary in order to avoid dilution. The result, however, will be and has been failure, for the reason that control means prohibitive mining costs.

Wholesale Mining.—Wholesale mining has to be viewed entirely in terms of the effects of dilution, for no low-cost system can be worked without this inevitable result. Due consideration must be given to the amount of dilution, its effect on the mill feed, and what means, if any, can be used to overcome this effect. In order to determine the effect of dilution on the value of the ore mined, it is necessary to know the value of the ore and the value and amount of dilution. Knowledge as to the value of the ore in the Alaska Juneau mine has been derived from muck samples, moil samples, and mill returns; in addition to the mill returns from normal operations, many tests have been made, not only on ore from various parts of the mine but also from bands intervening between ore bands, and from waste. All this accumulation of data is sufficient for the determination of ore values, and the conclusions arrived at are supported by actual returns; the mill feed and mill returns have held up absolutely to what all data indicated.

Caving System Finally Adopted

In the neighboring Alaska Gastineau property the mine and mill operations were based on two assumptions: (1) that the ore bands were "continuous and uniform" and (2) that by a modified caving system of mining the stopes could be confined to a single ore band. Neither of these assumptions was correct, and the combined effect of irregularly distributed values and inclusions of low-grade material and waste reduced the mill feed so that the recovery instead of being $1.50 per ton, as estimated, was but 82 cents per ton, 7 cents of which was in silver and lead. This mill feed was too low to justify continuing even with a 28-cent milling cost, and operations ceased.

The caving system adopted for mining the Alaska Juneau ore accepted dilution, and the new mill has been so rebuilt as to avoid having an excess of waste rock submitted to the process of fine grinding at a ball-mill cost of 40 cents per ton. This is accomplished by sorting the ore from the waste at advantageous points in the mill; 53 per cent. of the rock trammed is rejected, and the remaining 47 per cent. is fine milled. The cost of sorting and disposing of waste is about 7 cents per ton trammed and the final mill feed has nearly twice the assay value of the ore trammed.

In developing the mining system adopted, consideration was given to all physical conditions surrounding the orebodies. Full advantage was taken of the fault system, consisting chiefly of Silver Bow

fault and Nugget Gulch fault together with a number of subparallel sympathetic faults, all being post mineral. Silver Bow fault cuts the ore at a horizontal angle of 53° and completely severs all the ore bands, dividing the mine into two parts, locally called the "north half" and the "south half." Nugget Gulch is a strike fault and for all practical purposes is the marker for the foot wall of the ore; its dip varies from

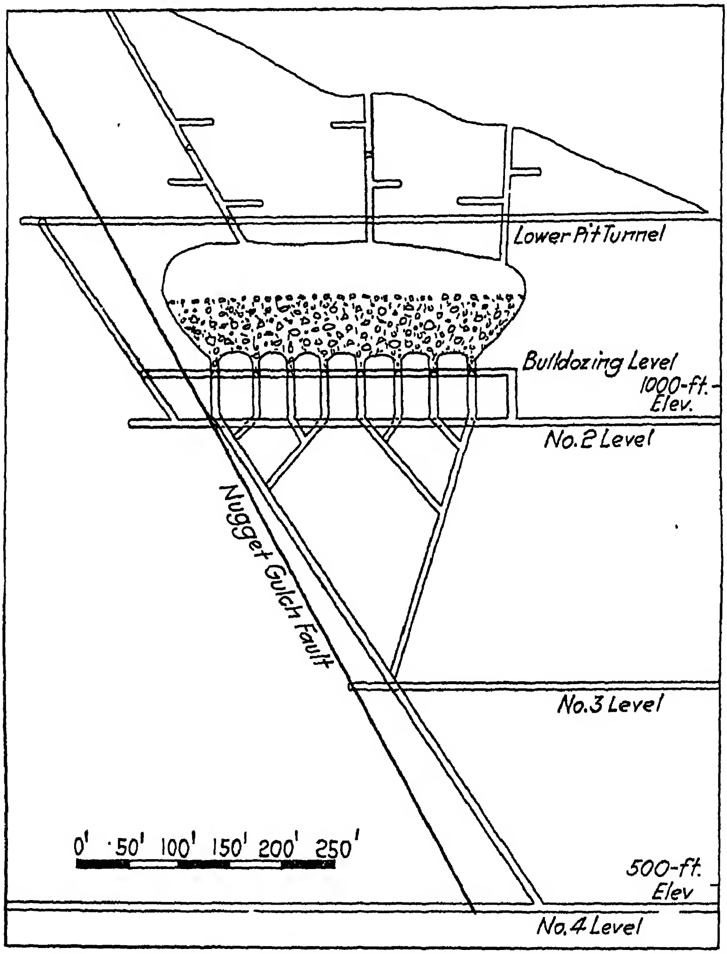

FIG. 1.—LONGITUDINAL SECTION THROUGH STOPE IN SOUTH HALF OF MINE.

55° to 60°. There is considerable fault breccia on each fault plane, and this sloughs away readily when undercut. The system as finally worked out consists in:

First, cutting out stopes of large area, so located that the foot wall of the stope is one of the faults.

Second, driving raises from the back of the cut-out stope area, through to the next two levels above.

Third, driving numerous powder drifts from and radial to these raises.

Fourth, provision of bulldozing chambers for drawing ore out of the stopes into the loading chute.

The first stope was cut out 750 by 250 ft. with the major axis parallel to the Silver Bow fault, and therefore 53° off the true strike of the ore. This undercut area is bounded on two sides by faults, each bearing the relation of a foot wall, on the third side by the continuation of the ore and on the fourth, or shortest side, by the hanging wall. As the stope extends upward it flares out on all four sides, and dilution enters only from the hanging wall or short side, and is thus reduced to a minimum.

The stope worked out exactly as planned, and has produced, to Nov. 1, 1923, about 2,200,000 tons. The dilution that will ultimately come into the stope in excessive quantities will be due to an expected gradual decline in values in the ore band as the back of the stope reaches higher elevations.

In the close vicinity of the stope just described two more stopes were cut out along the same general plan but with some differences in detail. These stopes were cut out parallel to the strike, with Nugget Gulch fault as their foot wall. Being rectangular, the same amount of hanging wall and foot wall were exposed; therefore, in order to avoid excessive dilution, these stopes were cut out smaller than the first stope and were made 200 by 240 ft., the shorter dimension being the strike measurement. An 80-ft. pillar was left between stopes and, as a further precaution against excessive dilution, the stopes were cut out narrower than the true width of the ore, so that the hanging side of the stope was not the hanging wall of the ore. Hence considerable caving has to take place from the hanging side before barren material begins to come in. The total production from these two stopes to Nov. 1, 1923, has been about 6,200,000 tons.

The two stopes described are in the north half of the mine; in the south half, two stopes have been prepared and put in operation, and preparatory work on a third is under way. The striking difference between the two halves of the mine is that in the south half the ore becomes higher in grade in the upper levels and in the north half it is expected to become lower grade in the upper levels. Therefore the stopes in the south half of the mine are located so as to start drawing the best grade of ore, in order to compensate for the expected declining values in the north half.

The stopes in the north are cut out 50 ft. above the main haulage level through which the ore is trammed to the mill, and the ore is handled directly from the stopes into cars. The stopes in the south half are 500 ft. above the main haulage level. Fig. 1 shows the general layout of the south stopes and the manner of drawing ore directly from the stopes into the oreway from which it is loaded into mine cars.

Preparatory Work and Stoping

The dimensions, shape, and location of the area that is to be prepared for stoping is selected after a careful consideration of the development work, assays, and all physical conditions; the two points of major importance are that the selected area has a fault for a foot wall, and that excessive dilution will be avoided. A high percentage of recovery of the total theoretical ore is at this time of secondary importance.

The first preparatory work consists of driving the haulage drifts under the area to be stoped and the intermediate, or bulldozing level,

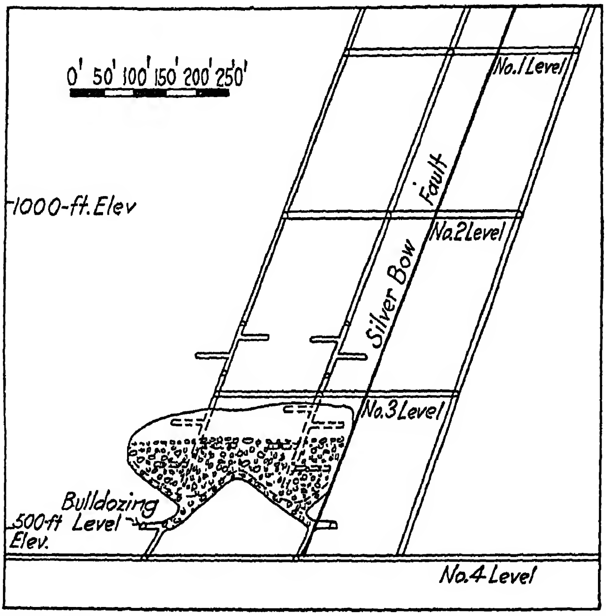

FIG. 2.—CROSS-SECTION THROUGH STOPE IN NORTH HALF OF MINE, SHOWING CONDITIONS SHORTLY AFTER COMPLETION OF PREPARATORY WORK, AND AFTER SEVERAL POWDER DRIFTS HAD BEEN BLASTED.

50 ft. vertically above the haulage drifts. The haulage drifts and intermediate level are then connected by chute raises, the chutes are built, and thereafter all muck from preparatory work is cheaply handled. The next step is undercutting the area to be stoped; all cutting out is done at the angle of repose so that muck will flow into the chute raises. The vertical cross-section of a cut-out stope, from foot wall to hanging wall takes the form of the letter W, see Fig. 2. During the preparation of the stope numerous small pillars are left to support the back. At the completion of the preparatory work these are blasted out, either by a large number of drill holes or by small powder drifts. Before the cutting out is completed, a number of raises are driven through from the back of the cut-out area to the levels above, spaced according to the character

of the ground. From these raises a number of powder drifts or coyote holes are driven in all directions, and after the preparatory work has been completed and stoping is started, the lower tier of these powder drifts is loaded and blasted. The drifts are spaced to give each one a burden of about 35 ft. of ground. After stoping is started, no one goes into the open stopes; access to the powder drifts for loading is gained from the levels above.

All preparatory work is done by contract, the contractor supplying all direct labor and the explosives, the company supplying all else. Prices vary with the character of ground, distances to be driven and condition of the labor market. The contract figures, which are approximately 60 per cent. of the total costs, range as follows:

	Per Foot
Main haulage drifts	\$10.00–\$13.00
Intermediate level	5.50– 8.00
Cross cuts	6.00– 8.50
Chute raises	7.00– 9.00
Cut-out raises	6.25– 7.50
Cutting out (per sq. ft., slope measurement)	0.35– 0.40
Stope raises	6.00– 8.00
Powder drifts	4.50– 6.25

Under the contract system the miners make a higher wage than otherwise, but there is a corresponding benefit to the company in a reduced unit cost of the work.

Up to Nov. 1, 1923, 146 powder drifts were blasted, consuming 490,900 lb. of powder or an average of 3369 lb. per blast. The largest amount used at any one blast was 8200 lb. No evil effects have resulted from these large blasts, and the work has been uniformly successful. The blasts bring down as much as 50,000 tons, but it is not always possible to determine the actual effect. During the time this system has been used, 8,922,500 tons have been mined, and the difference between that amount and the amount actually blasted is the result of natural caving. Generally, the gouge along the foot-wall fault will cave higher than the stope back; this makes a "cut" to aid the powder-drift blasts near the foot wall, which after being blasted makes a "cut" to assist in blasting the drifts along the hanging wall.

Bulldozing, Loading and Tramming

After the ore has broken away from the stope back, either by blasting or natural caving, it finds its way to the bulldozing chambers. These are at the top of the chute raises and there the large pieces of ore are broken sufficiently for all subsequent handling. In each bulldozing chamber is a grizzly made of 10 by 14-in. fir timbers capped on top with three lengths of 100-lb steel rail, two right side up and one between these two upside down, the separate timbers being spaced for 2-ft. clear-

ance. The grizzly has an inclination of about 10°; this permits a large rock to skid the entire length of the grizzly and avoids shock caused by the sudden stopping of large boulders. More recently, 99-lb., 15-in. standard H girders have been used for grizzly bars, with every promise of better service than the combined timber and railbar. All rocks too large to go through the grizzly are drilled with jackhamers and blasted. Those too large to come through the throat of the bulldozing chamber are drilled with stoper drills and blasted. The duty of each man on bulldozing ranges from 150 to 230 tons per shift, according to the size of the ore.

After passing the grizzly, the ore falls into the chute raises, which hold 200 tons each. From these raises, it is drawn through the chutes, which are equipped with an underswung arc-type gate, into 10-ton cars. The chute opening is 6 ft. long by 3 ft. high. The bulldozing chambers and chutes are spaced at 42-ft. centers, so that when a train is in the loading drift every third car is under a chute.

Bulldozing, loading and tramming are done by contract; the price prior to 1922 was 12 cents per ton, but in 1923 this was increased to 13 cents. For this figure the contractor provides all direct labor and the explosives. The amount that can be earned by the contractors is limited, anything between this amount and a certain sum is pro-rated among the rest of the bulldozing, loading and tramming crew who have a fixed base wage; any surplus above this latter sum is placed in a reserve fund and divided at the end of the six months' period between those who worked throughout the period.

The ore is trammed to the mill, 2 miles away, in trains of 44 cars hauled by 18-ton articulated Baldwin-Westinghouse locomotives over a 30-in. gage track of 50-lb. steel rail.

Costs and Other Data

The mining costs per ton trammed during the year 1922 and the firs ten months of 1923 are given in Table 3. To these figures, there should be added the mine's proportion of the overhead and general expense, the total of which per ton trammed was $0.0171 in 1922 and $0.0195 in 1923.

The charge for prepaid mining is the estimated figure necessary to extinguish the total mine preparatory work when the tonnage that has been and will be prepared for mining becomes exhausted. In 1922, certain work was started that was relatively more expensive per ton prepared for mining, and accordingly the figure was raised from 1 cent in 1922 to 2 cents in 1923. If all the preparatory work done up to Nov. 1, 1923, were charged to the total tons trammed, the mining costs to that date would be just under 33 cents per ton, not taking into consideration the several million tons now in the stopes ready for drawing.

TABLE 3.—*Costs per Ton Trammed, Alaska Juneau Mine*

	Labor	Power	Explosives	Supplies	Total
1922					
Prepaid mining	$0.0057	$0.0004	$0.0023	$0.0015	$0.0100
Stoping	0.0011	0.0002	0.0049	0.0003	0.0065
Bulldozing	0.0482	0.0047	0.0420	0.0077	0.1026
Tramming	0.0538	0.0026	0.0001	0.0204	0 0768
Total	$0.1089	$0.0079	$0.0493	$0.0299	$0.1960
1923*					
Prepaid mining	$0.0131	$0.0006	$0.0035	$0.0028	$0.0200
Stoping	0.0018	0.0001	0.0121	0.0012	0.0152
Bulldozing	0.0550	0.0062	0.0401	0.0085	0.1098
Tramming	0.0624	0.0038	0.0000	0.0155	0.0817
Total	$0.1323	$0.0107	$0.0557	$0.0280	$0.2267

* First 10 months.

TABLE 4.—*Unit Costs, Alaska Juneau Mine*

	Cost per Man per Day	Tons per Man per Day	Powder per Ton Trammed, Pounds	Av. Advance on Headings per Machine-drill Shift, Feet	Av. Total Cost Operating One Machine Drill One Shift
1922					
Development and preparatory mining	$15.31			2.07	$24.08
Bulldozing	16.68	161	0.24		
Loading and tramming	14.46	188			
Entire mine crew	10.90	49			
1923*					
Development and preparatory mining	$15.61			2.25	$26.69
Bulldozing	17.82	165	0.25		
Loading and tramming	13.47	176			
Entire mine crew	11.67	46			

* First 10 months.

The cost per man per day given in Table 4 is obtained by dividing the total shifts worked underground into the total expense underground. This figure for the entire mine crew during the years 1914, 1915 and 1916

was \$6.30, \$6.39, and \$6.31, respectively. A comparison of these figures with the corresponding figures for the years 1922 and 1923 shows at once the difference between present and pre-war conditions.

DISCUSSION

R. M. Raymond, New York, N. Y.—How has the earlier sampling checked out with the later operations?

A. H. Rogers, Brookline, Mass.—The early operations gave much better checks than later on account of the more selective method of mining used. When the Gastineau mine started operating, three stopes were worked according to this system. A slice was carried up by breaking out the ground at the ends, which would be sufficient to cave the back. The ground is very loose. As a matter of fact, later, in all the Gastineau stopes, it was never necessary to go into the stopes at all; the back caved continuously so that ore could be drawn at the chutes. All this ore would cost on the car was about five cents per ton, this being the cost of bulldozing and pulling out of the chutes.

Enoch Perkins, Wharton, N. J.—At the Perseverance, later, there were three classes of stopes; in slate, metagabbro and slate, and schist. The slate, of course, gave the most trouble from caving. The stoping costs in the schist stopes, where it was usually necessary to break the ore from foot wall to hanging wall, were much higher than in the slate stopes, which often required only undercutting with occasional slices along the foot wall, after which the ground caved of its own weight. In an attempt to control the caving into the hanging wall, the stopes on the lower levels were laid out much shorter than those above.

One of the most interesting features, to me at least, of the Alaska Juneau operations, is the remarkably low tramming and bulldozing costs when the character of the broken ore is considered. Naturally, due to the mining method, but little ore is drilled and blasted in the stope. The major portion is caved by means of the raises and blasting drifts shown, which undoubtedly provides, at times, exceedingly blocky broken ore for tramming. The fact that between 250 and 300 tons per bulldozer are put through the grizzlies seems, therefore, to be a remarkably good operation.

R. M. Raymond.—How does the general hardness compare with other materials?

Enoch Perkins.—The metagabbro may be compared with a fairly close-grained granite. The schist, being highly silicified, was much more difficult to drill and break than the gabbro. The schist broke very blocky, which increased bulldozing costs. The slate is usually a typical black graphite slate that drilled easily and caved readily.

I have heard that Alaska Juneau is going into Ebner ground. If it does, more metagabbro will be met than is the case in the present stopes. The stoping methods will, therefore, need to be modified, which might result in greater costs along with higher mill heads. The old Ebner surface workings were entirely in metagabbro, as were the glory-hole operations years ago of the original Alaska Juneau Co. This portion of the lode contained, on the average, much better values than the slates now being mined, and there should be less dilution on account of the more substantial hanging wall.

R. M. RAYMOND.—Was any attempt made, in the Gastineau, to increase the amount sorted out?

ENOCH PERKINS.—It was impossible with their mining methods. The three stopes Mr. Rogers mentioned turned out to be the richest stopes in the mine. In the mill runs of the old Perseverance mill, all the ore came from these stopes and there was but little trouble from dilution for the mining was not carried far enough for caving to commence.

H. G. MOULTON, New York, N. Y.—The mining method at Juneau involves caving only in an incidental way. As a matter of fact it is a form of overhand stoping, and the caving is not the result of undercutting the orebody but is incidental to the large scale of the operation. The most interesting feature of the method is the use of powder drifts instead of drill holes for breaking down the backs of the stopes.

In determining the grade of the ore, the presence of quartz is probably an aid in the judgment of values. The determination of grade underground by inspection and judgment is also applied on the surface in sorting out the less valuable slate and milling only the higher grade ore. By sorting out waste before milling, the cost of milling per ton of ore mined is reduced by an amount greater than the net value of the material discarded.

A. H. ROGERS.—It is a question whether the sorting pays in connection with the milling. Mr. Bradley says it costs 7 cents per ton trammed. The milling cost at the Perseverance was about 25 cents. The first operation involved crushing and roughing out a lot of this slate on Garfield tables; the cost of the roughing operation certainly did not exceed 7 cents, and very likely it was less. In that roughing operation, they eliminated three-quarters of the ore, as I remember it.

H. G. MOULTON.—The problem is an economic one. With an increased output, the costs are lowered. The method in use at Juneau involves the increase of tonnage mined so as to produce minimum costs, the rejection by hand sorting of a large part of the material mined so as to reduce the total cost of milling, and the acceptance of a certain loss of gold value in the material rejected by hand sorting. The investment

in milling facilities is one element in the problem, as the carrying charges on any additional investment necessarily form part of the costs that would be incurred if the material rejected in hand sorting were to be milled.

P. R. Bradley.—My inference from Mr. Rogers' statement that "it is a question whether the sorting pays in connection with the milling" is that he is comparing sorting in the Alaska Juneau mill with roughing in the Alaska Gastineau mill. If Mr. Rogers were familiar with the present Alaska Juneau practice he could not make such a comparison and come to the conclusion quoted.

In the Alaska Juneau mill, sorting is the first operation after coarse crushing. If Mr. Rogers would have the waste, instead of being sorted out at this point, go on through the mill to be rejected by roughing, it would first have to be fine-milled at a cost of 26 cents per ton; and, furthermore, it would displace from the fine-grinding department of the mill, which is running to full capacity, mill feed having nearly twice the gold assay value of the rock trammed.

Top-slicing

By means of top-slicing methods, wide veins, masses, or thick beds of soft ore may be mined, where the caving of the overburden is of no consequence. The ore is mined in a series of horizontal floors or slices, taken in descending order, from the top of the deposit. Each floor is mined in small sections, the roof of each section being allowed or forced to cave before an adjacent section is attacked. The method as used in two mining districts—Marquette, iron, and Pachuca, silver-gold—is described in the two papers following:

Mining Methods of Marquette District, Michigan

By S. R. Elliott, J. E. Jopling, R. J. Chenneour, E. L. Derby

(New York Meeting, February, 1923)

The Marquette range, on which are situated the iron mines of Marquette County, together with a few in Baraga County, Mich., extends from a point 10 miles southwest of Marquette westward for 30 miles. The tracts are usually a multiple of the standard 40-acre parcel, which is the smallest government subdivision of a square mile or section.

About half of the mines are held in fee, these being owned by the older mining companies. Some of them date back to about 1880, and a few are as early as 1850. During the past 40 years most of the mines opened have been on leased lands, the royalty being either for a stated amount per ton or a percentage of the selling price of the ore at the lower lake ports, or the selling price on board of cars at the mine.

The first merchantable body of ore discovered in the Lake Superior district was found, in 1845, at what is known as the Jackson mine, near Negaunee. This ore was hard hematite. After unprofitable attempts had been made, in 1848, to smelt it in forges, shipments were begun to lower lake ports, which increased rapidly upon the completion of the locks at Sault Ste. Marie, in 1855, and the railroad from the mines to Marquette, in 1857.

Beginning with the Pioneer, in 1858, a number of small charcoal furnaces were built to smelt a part of the product. At various points in the upper and lower peninsulas, large charcoal furnaces are still making iron from the ores of the Marquette range. In connection with these furnaces are byproduct plants.

As the deposits were opened up, soft ore was encountered, but for a few years this was disregarded, as only the hard ore was used. Underground mining was begun about 1880 and during the next few years the open-pit mines producing high-grade ore were exhausted.

The first mines in the district were owned by the Jackson, Cleveland, Lake Superior, Lake Angeline, Champion, Iron Cliffs, Humboldt, Republic, and Michigamme companies. Many of these companies have been merged in the holdings of The Cleveland-Cliffs Iron Co., which controls most of the tonnage of the district.

Ore shipments are made from Marquette through the docks of the Duluth, South Shore & Atlantic railway, constructed in 1857, and from the Lake Superior & Ishpeming railway, constructed in 1896. A portion of the tonnage is also shipped over Chicago & North Western to Escanaba. Since the above dates, larger and more modern docks have been built.

The average number of men employed in the district for 25 years is 4215. The average annual shipments from 1902 to 1920, inclusive, have been 3,827,659 tons; the total shipment to the end of 1921 is 137,237,513 tons.

Geology of District

The geology of the Marquette range is described, in Monograph 52 of the U. S. Geological Survey, by Van Hise and Leith. The iron formations occur in the Huronian series of the Algonquin group of pre-Cambrian rocks. The sedimentaries, in which occur the principal mines, stretch from Marquette through Negaunee, Ishpeming, and Champion to Michigamme and Republic, with a separate area at Gwinn. The series consists mostly of quartzites and slates, interbedded with them being the jaspers of the iron formations. All of these rocks are faulted and folded and are crossed by dikes of greenstone or diorite. The Negaunee iron formation, or jasper, in which most of the mines occur, is a combination of iron oxide and silica containing, according to the U. S. Geological Survey, about 29 per cent. iron. No commercial use has been made of this material. Its greatest thickness, where proved by drilling at Negaunee, is over 2000 ft. It rests upon slates belonging to the formation, which is locally known as the Siamo, and is overlaid by quartzites or slates of the Goodrich formation. The soft ores occur as secondary concentrates either near the base or near the top of the jasper, the latter resting upon interbedded diorite intrusions. These orebodies are often limited in depth by dikes or faulted portions of diorite or slate, which serve as impervious bases on which concentration has taken place.

The hard ores occur only at the top of the Negaunee formation, being underlaid by a few hundred feet of hard-ore jasper, which again lies on the jasper of the soft-ore formation. The hanging wall of the hard ores is quartzite or slate.

The Michigamme slate formation, which overlies the upper quartzite and the Negaunee formation, contains interbedded iron formations, which in places produce limonite ores.

The Gwinn, or Swanzy, subdistrict of Marquette range is placed, by the U. S. Geological Survey, in the Michigamme formation. Here the ore is a soft hematite, found at the base of a jasper 100 ft. or more in thickness, and resting on comparatively thin beds of black slate and quartzite or arkos overlying the granite. The iron formation is overlaid by the slates of the Michigamme formation.

The physical structure of the ores on the Marquette range is excellent, none of them having a fine enough structure to be objectionable to furnace men. The hard ores are used in lump form in the open-hearth processes; they are crushed for use in the ordinary blast furnace.

Description and Topography

The district is about 800 ft. above Lake Superior, or 1400 ft. above the sea level. The surface is hilly and rocky. Lakes and swamps are bordered with terraces of glacial origin, above them rising rocky hills of iron formation, quartzite or diorite, the tops of which are from 50 to 200 ft. above the glacial terraces. Because of the proximity of Lake Superior the summer climate is cool. The prevailing northwest winds bring heavy snow falls from the beginning of November until the middle of April. The winter temperature is modified by the lake, which never freezes over entirely. The average yearly rain fall, which includes the equivalent in snow, is about 32 inches.

Mining timber is brought in by rail. Practically all the white pine of the Upper Peninsula has been removed, and in the neighborhood of the mines, most of the hardwood also has been cut. There still remain districts where there are large stands of hardwood, together with tamarack, hemlock, and cedar.

Labor conditions have been excellent. The mining population, derived chiefly from northwestern Europe, has been industrious and thrifty. Most of the men own their homes, the lands upon which they stand being either purchased or leased from the companies on easy terms, which induce building. However, since the war, many workmen have been attracted by high wages to the large cities, this movement being accelerated when the mines were shut down during the recent depression.

The ore is transported by rail from the mines to the docks in hopper-bottomed cars of 50-ton capacity. It is there dumped into pockets of 200-ton capacity. In loading vessels, the ore is delivered through spouts, which are lowered to the hatches. These spouts are 12 ft. from center to center, and the hatches on the boats are 12 ft., 24 ft., or a multiple of 12 ft. apart.

Pumping at the mines varies from a few hundred to about 3000 gal. per min. The overlying sand and gravel are often saturated with water, but owing to embedded clays and other courses, this is seldom drained until broken by the extraction of the ore, under the caving system.

Exploring

The earliest explorations were for the purpose of finding the ledge or deposit from which had come the many broken masses of hard ore found

lying upon the surface or in the glacial material. Succeeding explorations were conducted by test pitting through the overburden, drilling the same way from outcrops, or tunneling into the rocks themselves. In many cases, shallow shafts were sunk, from which drifts were driven.

The diamond drill was used, as early as 1869, for deep holes in hard orebodies, but its use was not customary until about 1878; since then it has become the usual method of exploring for both surface and underground work. The churn drill has not been used much for exploring, owing to the hardness of the rock capping to be penetrated.

Sampling

As all explorations of orebodies at present are by diamond drilling, the only sampling is that of the core and sludge, from the drill holes. These are collected after each 5-ft. run and later analyzed; the weighted average of the two, figured on the proportion of each covered by the length of the run, is the analysis for that run. These analyses, combined for the entire hole, give the depth and grade of known ore encountered. Duplicate samples of core and sludge from each run are preserved for reference. If the presence of soluble sulfur is suspected, the amount of water pumped down the hole and the amount coming out are measured, and samples of this water taken at definite intervals, these samples being analyzed for sulfur. This is then combined with the analysis of insoluble sulfur in the core and sludge. When drilling through hard ore, almost complete recovery of core can be made; while in soft ore practically no core is obtained, and the only analysis is that of the sludge. The sludge is collected by causing the water from the drill holes to flow into boxes 4½ by 1½ ft., with two baffle boards, in which the particles of ore held in suspension settle out.

Estimating

The methods of estimating are those customarily used in the Lake Superior district, both for ore found by diamond drilling and for developed ore underground. This is a comprehensive subject and should be treated in a separate paper. Plans and cross-sections are made both of explorations and mine workings to show the area and depth of the deposits as soon as ascertained. The limits of formation are shown on these drawings as a result of careful geologic examination. Considerable latitude for judgment must be permitted in the case of partly developed orebodies, which for the purpose of estimating the cost of development must be divided into ore in sight and prospective ore. The number of cubic feet per ton varies from 8 to 9 in hard ores and is about 12 in soft hematites,

while for limonites it is as high as 13 or 14. The U. S. Geological Survey states that the soft ores of the Marquette range average 12 cu. ft. to the ton. This is borne out by a series of careful tests, which have recently been made in several mines.

Accuracy of Method

Due to the irregular shape of the various orebodies, large discrepancies have often been found between careful estimates based on exploration and the tonnage eventually recovered. Drill holes in some cases follow chimneys, the formation proving barren except for a small cross-section in the vicinity of the hole. On the other hand, some deposits have proved to be much greater than estimated by drill holes. Reasonably correct estimates can be made in shallow deposits from a number of short holes at close intervals, but the difficulty increases for depths over a few hundred feet, due to the deviation of the diamond-drill holes and the lack of knowledge of the geology of the formation.

In the case of shallow regular orebodies, as on the Mesaba range, the tonnage can be accurately estimated and the percentage of extraction determined. This is not the case on the Marquette range, where the ore deposits, as a rule, are deep and extremely irregular.

History of Principal Mining Methods

The early shipments of iron ore previous to 1860 were made largely from loose masses found scattered on the surface. After this source was exhausted, open pits were developed, some of which continued to operate until about 1880. Drilling by hand, blasting with black powder, loading into carts drawn by horses or mules, and again loading into 7-ton cars on the railroad constituted the usual method. Shafts were started for collecting the water in the pits, so that it could be pumped. As the pits grew deeper and available ore seams were followed, winzes or stopes were sunk, the ore being raised by horse whims. Most of the early appliances were introduced from Cornwall, whence the first miners came. Details of the mining methods, including cost, are given in Volume 1, Geological Survey of Michigan, published in 1873.

As the open pits were continued, they increased in depth until a point was reached, on account of the ore dipping under the rocks, where it was not profitable to remove the overburden. It was then necessary to sink inclines and provide mechanical means of hoisting. The hard ore was then mined in open stopes, pillars being left to support the capping.

From 1875 to 1880, much of the mechanical equipment was introduced, including rock drills, electric lights, and electric signals. At this

time dynamite replaced black powder as the chief explosive. Hard-ore mining continued in about the same manner at depth. The breast stoping method of room and pillar was used, in which only enough ore was left in pillars to support the hanging.

The first soft ore was found near hard ore deposits. The demand for this class of ore was limited, previous to 1880. It was mined in open pits, but when it became unprofitable (on account of the increased depth and the dipping of the ore under the rock, to remove the overburden) underground mining was started. This soft ore could not be mined by the open stope and pillar method; timber was necessary to keep the places open. The principal method of mining soft ore was, in the middle eighties, almost entirely by the square-set system of rooms and pillars. The rooms were usually three sets, or 21 ft., wide and as long as the orebody. As a rule, the pillars were of the same width as the rooms. In many instances the rooms were carried to a height of ten or twelve sets, or 70 or 84 ft. After the ore had been mined in the rooms, an attempt was made to get what was left in the pillars by raising and running them. This system was extremely wasteful, as the ore soon became mixed with rock and the grade lowered so that work had to be stopped.

The caving system was introduced by miners from the north of England, where it originated. The method there was to mine from a sublevel immediately below a mat of timber, which was kept propped up until the retreat of mining began. Every effort was made to maintain this timber mat, for when it was destroyed a new mat had to be made at great cost. Modifications of this caving system were introduced during the early eighties, until it became the accepted method about 1887 to 1890 (see J. P. Channing in the Lake Superior Mining Institute, Volume 19); the introduction of the caving system was hastened by the decreasing local supply of large timber.

Permanent shafts in the foot wall were rare before 1895. Timber was used exclusively for shafts, shaft houses, and trestles. About 1900, steel replaced wood in shaft houses and, about 1910, concrete and steel were used for shaft lining. The most recent practice, inaugurated in 1919, is to build enclosed shaft houses of reinforced concrete. Electric haulage was introduced in 1892, since which time its use has become general.

Certain changes in mining conditions were brought about by legal restrictions, though in justice to the mining companies it should be said that much of the beneficial legislation enacted was prompted by the managers. The election of the mine inspector in each county, provided for in 1887, assured a greater degree of care for the safety of the workmen, which was increased after the passage of Michigan's workmen's compensation law in 1912. All large companies now have a department for safety inspection and first-aid training, although these are not required by law.

Early operations were wasteful, because of the lack of system and the necessity of marketing only the higher grades of ore. As the number of leased properties increased, it became necessary that all ores should be taken out in a workmanlike manner and unnecessary waste prevented. The principal causes of loss in the early years of the district were: (1) the lack of preliminary exploration and the beginning of caving before the limits of the orebodies had been determined; (2) the fact that softer material could be mined more cheaply than hard, which consequently was left in place; (3) in attempting to reduce the cost too great a vertical distance was taken between sublevels; (4) the lack of proper maps and systematic method of laying out the work.

The larger and more progressive mining companies, realizing these mistakes, inaugurated geological investigation, systematic development and close supervision, which resulted in the present methods by which losses have been reduced to a minimum.

Marketing

The demand for crushed soft ores for charcoal furnaces necessitated the introduction of crushers in shaft houses.

Attempts have been made to concentrate some of the lean ores but, principally because of the intimate association of silica with the iron oxide, these have failed. The first concentrating plant in the district was built at the Jackson mine in about 1880. It failed because jaw crushers and rolls could not be made hard enough to withstand the wear and tear (manganese and other alloys of steel were not then in general use) and because, regardless of how fine the crushing might be, the particles of iron oxide and silica were too closely associated to be separated. Magnetic concentration was attempted, about 1890, by Thomas A. Edison at Humboldt and Michigamme by means of machinery introduced from Sweden for the magnetic treatment of magnetites. These attempts failed, owing to the small amount of ore available for concentration. At the American-Boston, a concentrator for soft ores was used until the mine was shut down. This method depended on a special structure of low-grade ore. Crowell & Murray's "Iron Ores of Lake Superior" gives valuable information on the various ores of this range.

Mining Methods in Use

There are no rich ores close enough to the surface to permit open pit mining. Doubtless many deposits, now exhausted, that were opened underground could have been stripped with modern equipment and the ore mined at a profit. Only the lean ores are now mined in open pits.

The factor deciding the question of open pit or underground mining is the cost of stripping the overburden as compared with underground cost. Climatic conditions do not interfere, as shipments are made only in the summer. In open-pit mining, the systems used are: steam shoveling directly into standard railway or narrow-gage cars; milling into raises to underground drifts, then tramming the ore to a shaft, where it is dumped, hoisted, and run into ore cars.

Varying conditions necessitate a number of minor differences in mining methods, because hardly any two orebodies in the district are of the same size, shape, or physical structure. Underground mining may be separated into hard- and soft-ore mining. The hard ores are comparatively unimportant, as only a few mines on this range contain this grade. The systems used are either breast stoping into rooms and pillars, or shrinkage stoping if the vein is narrow, steeply dipping and has firm hanging and foot walls.

Underground Mines

Most of the soft ore mined on the Marquette range is won by the top-slicing method. The typical orebody is a large mass with width exceeding thickness, lying in a basin of slate or diorite, or both, with a flat pitch and with the overlying jasper for a hanging wall or capping. The width may be as great as 1200 ft., the thickness 200 ft., and the length indefinite. In orebodies of such shape and dimension, top slicing is the only system by which great loss of ore can be avoided and the ore obtained without the grade being seriously affected by its being mixed with rock. The top-slicing system is flexible, in that any horses of jasper or large dikes can be left. By daily sampling from the breast of the working places, the ore can be hoisted and shipped, or stocked, according to grade. In large orebodies, slicing is carried on at different elevations, as extremely large sublevels are practically impossible to keep open. A large product can soon be obtained by starting work in a number of different places, each of which must be immediately below the hanging jasper. The disadvantage of this system is the amount of timber required and the heat generated by the decay of the timber. However, as practically all of the mines have two openings to the surface, good ventilation can be obtained by natural or mechanical means and the temperature kept within reasonable limits.

Mine Opening

Mine openings are entirely by shafts, all modern ones being vertical; the deepest in the district is about 2500 ft. The size regarded as standard is 10 ft. 10 in. by 14 ft. 10 in. inside. This is divided into four compartments, the arrangement of which is shown on Fig. 1. Modern shafts are

constructed with concrete walls and steel sets; some are circular, others rectangular. Great care is taken to locate them in the solid foot wall at a point where they will not be disturbed by caving.

The arrangement of the loading and discharging pockets is shown in Fig. 2. There are usually three 60-ton storage pockets, from which the ore is drawn into two measuring pockets, each holding enough for one skip. In some cases, additional storage is gained by raises from one level

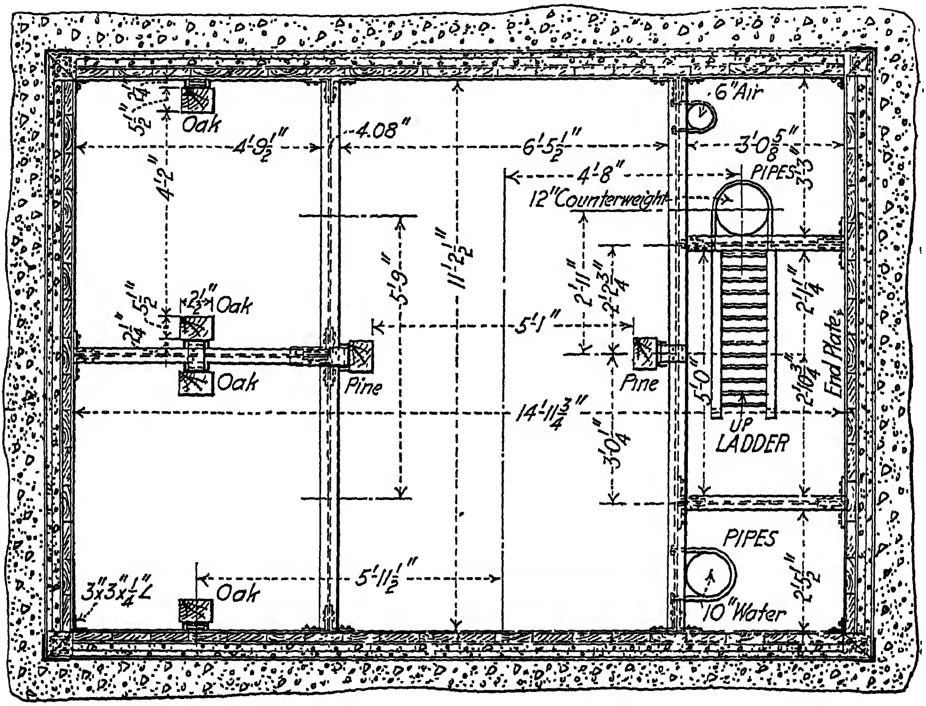

FIG. 1.—PLAN OF NO. 1 SHAFT.

to another, thus avoiding the expense of installing pockets on each level. In some mines storage raises at the shafts, 200 ft. high, are satisfactory.

Underground Development Plans

In Fig. 3 are shown the main levels, sublevels, raises, and chutes, with their relative dimensions and intervals.

Mining

Drilling and Blasting.—In drifting through hard rock, water-feed hammer drills on cradles are used, while for drifting and raising in ore hand machines of the jackhamer type, fitted with auger bits, are employed. For rock raising, the stoper is in common use. For shaft-

sinking, air or water-feed hand sinking machines are used. The size and shape of the steel and bits are shown in Fig. 4; Fig. 5 shows the arrangement and depth of holes for cuts in ore and rock drifts. For tamping,

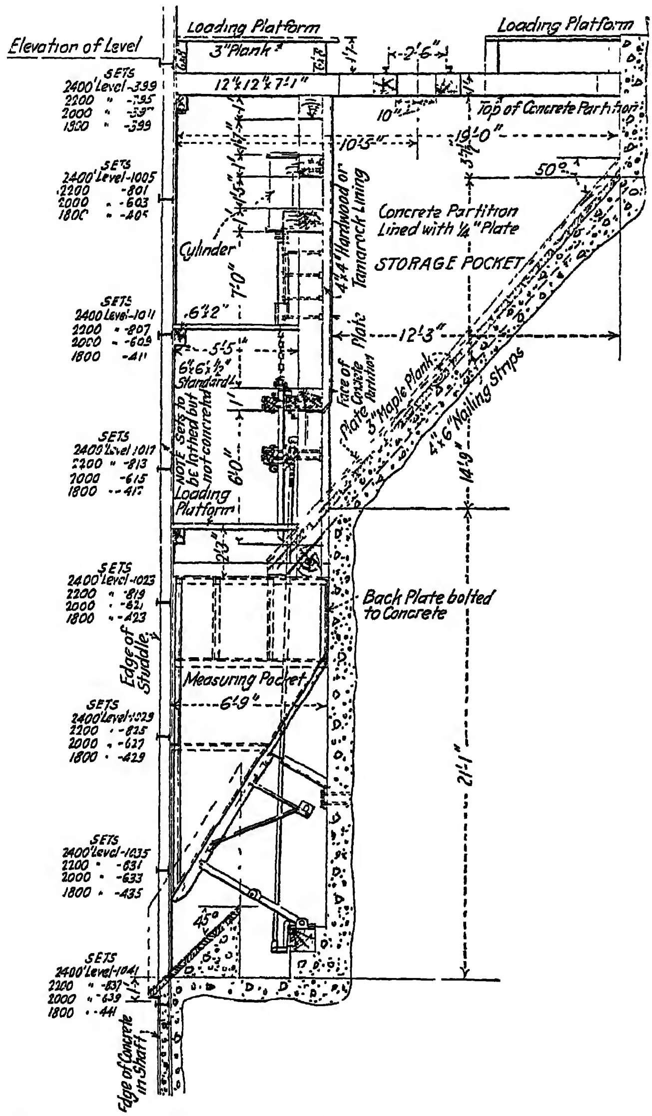

FIG. 2.—ARRANGEMENT OF LOADING AND DISCHARGING POCKETS.

paper bags filled with fine material are supplied. In most soft ores, the explosive is 40 and 50 per cent. low-freezing ammonia; while in some of the harder ores 50 and 60 per cent. gelatine is used, occasionally 80 per cent.

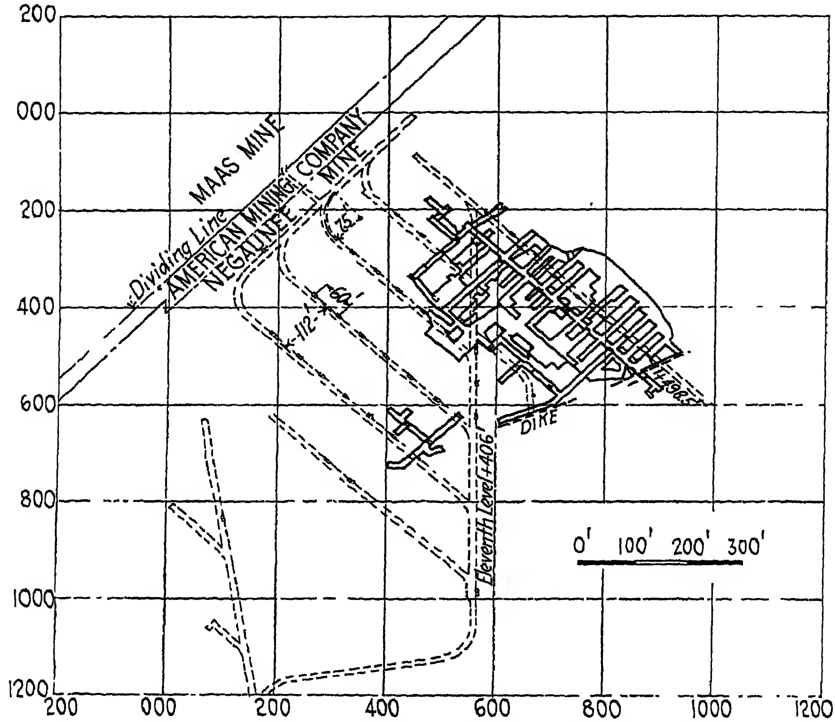

FIG. 3.—500-FT. SUBLEVEL OF NEGAUNEE MINE.

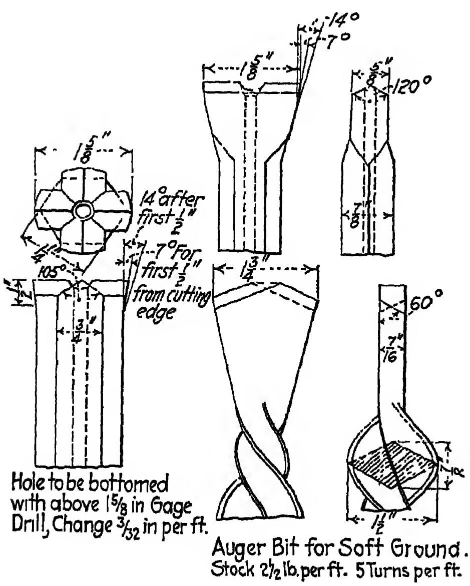

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FIG. 4.—FORMS OF DRILL BITS USED.

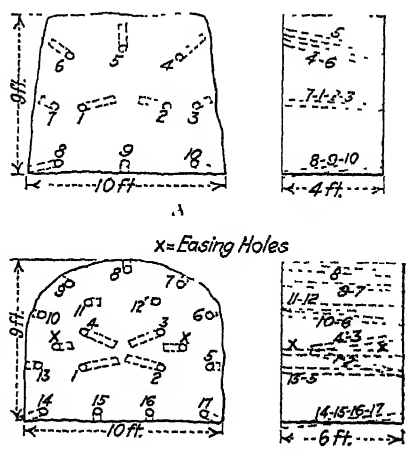

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FIG. 5.—DRILL ROUNDS USED AT MAAS MINE.

gelatine is necessary. The air pressure at the drill is usually between 70 and 80 pounds.

Drifting and Stoping.—In laying out the main haulage levels, parallel crosscuts at intervals of 150 ft. are driven. From these crosscuts, raises are put up at intervals of 40 to 60 ft.; these raises are carried through to the capping. At the top of the raises, immediately below the jasper, sublevels are started. Crosscuts are driven to the proper limit and slicing is commenced. As the ore is removed, the floors are well covered with either lagging or ⅝-in. covering down boards. Succeeding sublevels are driven at intervals of from 10 to 12 ft. On each sublevel, during the process of slicing, the floors are well covered. Fig. 3 gives details for position of drifts and raises.

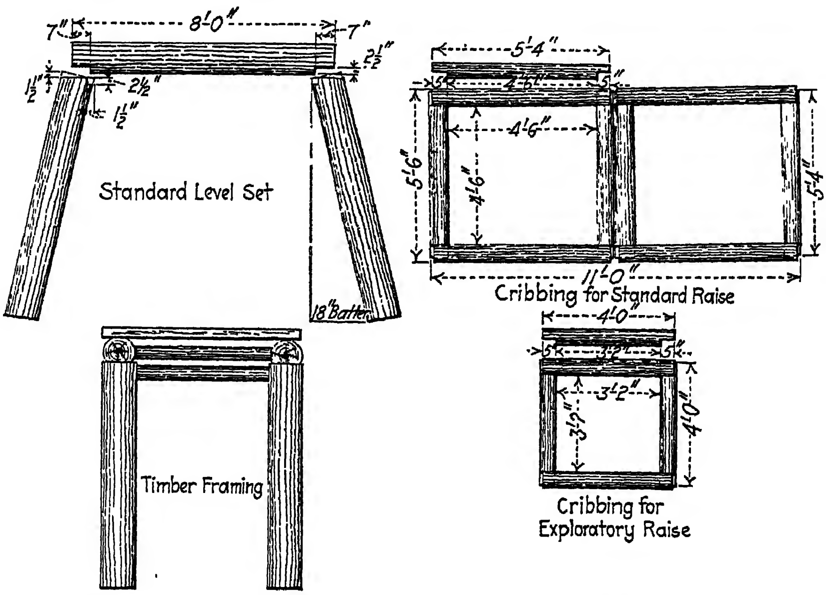

FIG. 6.—DETAILS OF TIMBER FRAMING USED BY CLEVELAND-CLIFFS IRON CO.

Timbering.—In the main and sublevel drifts, round hardwood, hemlock, and tamarack timber are used. On this range, timber has not been chemically treated, although preparations are being made to do this. It is thought that before long all of the timber for main levels will be chemically treated.

Timber that has been framed on the surface is delivered at the bottom of raises on timber cars. From these points it is hoisted to the working places by means of small air hoists. It is not the practice, on this range, to remove any timber during the process of working the caving system. The details of the timber framing are given in Fig. 6.

The legs of the standard level set are usually 8 ft. long, though 9-ft. legs are used at the bottom of raises and in some haulage drifts. Logs

are delivered in 8-ft. and 16-ft. lengths, the 7-ft. stub from a 9-ft. leg being used as a short leg or cap in sublevels. For main-level sets, legs are 12 to 14 in. in diameter; for sublevels, 8 in. to 10 in. and 10 in. to 12 in. Sets are usually 5 ft. apart and are braced as shown. A longer brace, behind the lower one shown, is spiked to both legs. Lagging and, where necessary, blocking is placed above the caps.

Specifications for Timber

Stull Timber (Legs and Caps)

Hemlock, hard maple, soft maple, yellow birch, tamarack, and Norway pine. Must be sound, straight, and green. Ash, white birch, poplar and balm of gilead not accepted.

Tamarack in top diameters of 8 and 9 in. containing not more than ¼ in. sap rot in depth, accepted; in diameters of 10, 11, 12, and 13 in., containing not more than ½ in. sap rot in depth. Lengths, 8 and 16 ft.

Cribbing

Tamarack, spruce, pine, hemlock, maple, and yellow birch. Must be sound, straight and green. Top diameter, 6 to 8 in. Lengths, 5 ft. 4 in., 10 ft. 8 in., and 16 ft.

Lagging

Straight sound cedar, tamarack, and spruce (10 per cent. jack pine permitted).
Round, 3- to 4½-in. top, larger than 4½ in. to be split.
Split, not less than 2½ by 4 nor greater than 3 by 6.
In 5-ft. lengths, 160 cu. ft. per cord.
In 7- and 8-ft. lengths, 128 cu. ft. per cord; not less than 125 pieces.

Covering-down Boards

No. 3 maple and birch. To be resawed from 2-in. maple and birch hearts to ⅝ in. To be sound, green lumber.
Widths, 6 in. and wider.
Lengths, not under 6 ft. and not over 9 ft.

Underground Track Ties

4 ft. 6 in. long, 4½ in. thick, 4½-in. face.

Poles

10 ft. long, cut from 20- and 30-ft. lengths, 3 to 4½ in.

Underground Sampling.—All working places are sampled at intervals of about 5 ft.; these samples make it possible to grade the ore. In addition, the ore from each chute is sampled as the motor cars are filled; this gives a check sample on the grade. All cars, before they are dumped into pockets at the shaft, are sampled, the samples being kept separate, according to mine chutes. On the surface, during the shipping season, a sample is taken from each skip as the ore runs into the railroad car; during the stocking season, it is taken from each car before it is dumped on

the stock pile. The samples taken on the surface are the ones reported for the grade, those taken underground are simply used as a check.

Loading Machines and Scrapers.—Two heavy types of loading machines (the Shuveloder and the Hoar) have been successfully used in main level drifts. These machines are not practical, except on main levels. For sublevels, the John Mayne sublevel loader, Fig. 7, has been successfully used for over two years; this machine will be on the market in a short time. It is simple in construction and has few moving parts. It stands up under continuous work and can be operated by any miner. Records for a year for a gang using this loader show an increase over hand shoveling of 93.6 per cent. in tons per man per day; a decrease in

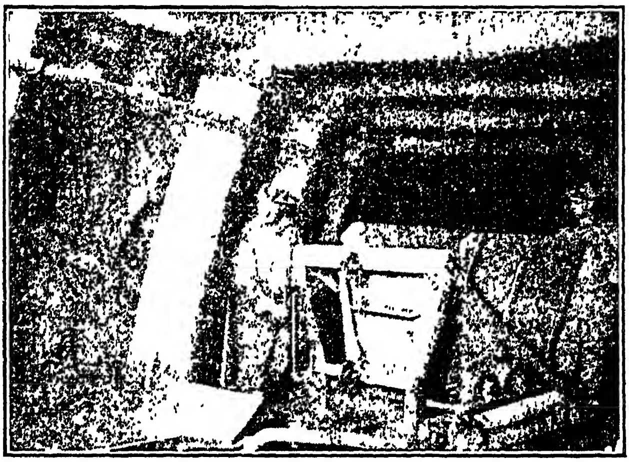

Fig. 7.—John Mayne sublevel loader.

price to contractors of 32.6 per cent., and an increase in monthly earnings of miners of 19.5 per cent. Scrapers operated by double-drum air hoists have been used successfully in a limited way in both hard and soft ores. They load rapidly in a straight drift up to 75 ft. from a raise.

Tramming and Haulage.—Electric haulage is used almost exclusively in the district. Direct current is generated from alternating by rotary converters, situated on the surface or underground. Locomotives are 6 tons in weight and the current is received from an overhead trolley. The ore cars are of steel, side dumping with saddle backs, of 64 cu. ft. capacity, or approximately 4 tons of ore. The gage of tracks is 30 in. and the weight of rail 30 to 40 lb. The grade is 0.5 per cent. with the load. The cars are usually equipped with roller-bearing wheels and are dumped by hand at the shaft though the most recent installations have been rotary dumps, using round bottom cars.

Hoisting.—At most of the mines electricity is used for hoisting, although some still use steam. These hoists vary in horsepower from

400 to 900, depending on the depth of the shaft and the size of the skips. As alternating-current hoists of this size throw a heavy variable load on the power line, the newer hoists are equipped with flywheel motor-generator sets following the Ilgner system. With the skip hoist operated by direct current, an approximately constant power load is

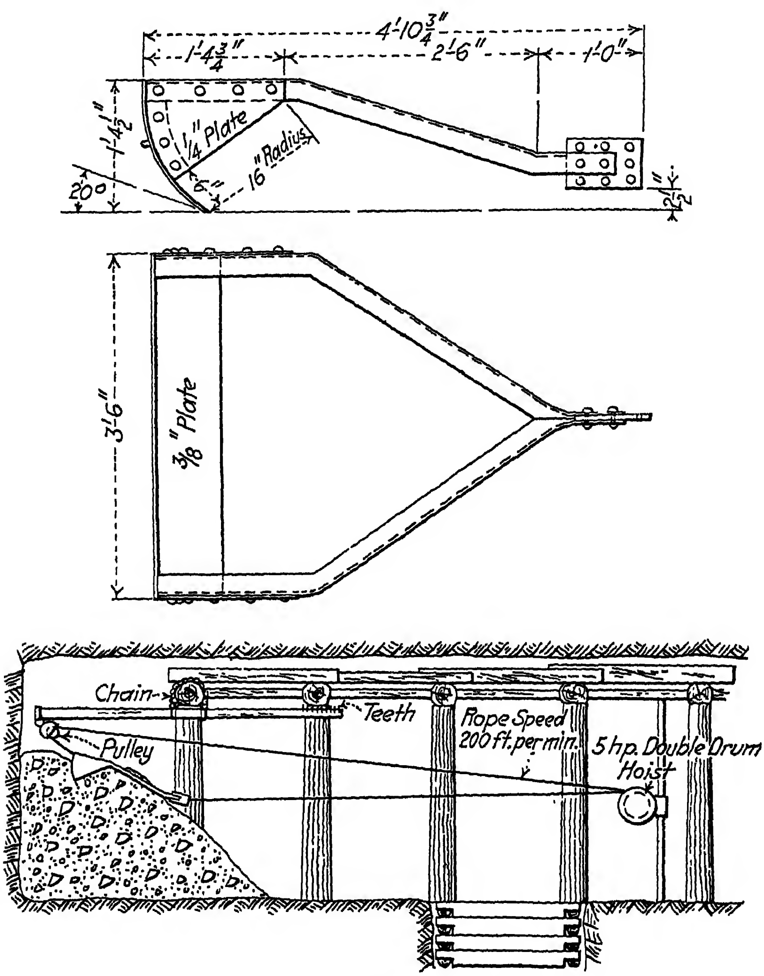

FIG. 8.—SCRAPER FOR SOFT ORE.

maintained. The drums are from 8 to 10 ft. in diameter and from 8- to 14-ft. face. Hoisting is usually done in balance, at a speed from 1000 to 2000 ft. a min. For a 4-ton skip, which is the average size, a 1¼-in. plow-steel rope is used, except in the deeper shafts, which use 1⅜-in. ropes. At almost all mines, a separate hoist is provided for the men. The cages have a capacity of from 24 to 30 men, and the speed,

when men are being handled, is about 800 ft. per min. The hoists are provided with Lillie overwind device and the cages are equipped with safety catches and are balanced by counterweights. The counterweights are cast-iron cylinders, operating in 12-in. iron pipe. Most of the head frames are of steel, though one mine has an enclosed concrete structure.

Pumping.—Both plunger and centrifugal pumps are used, the motive power usually being electricity. These have capacities from 500 to 1600 gal. per min. against heads from 500 to 2400 ft. The amount of water pumped varies, in the different mines, from 150,000 to 3,500,000 gal. per day.

Air Compression.—Most mines are equipped with two-stage intercooled compressors of a capacity of 2000 cu. ft. per min.

Ventilation.—Nearly all mines have two openings and sufficient ventilation is therefore provided by natural means. In a few cases, where there is only one shaft and the connecting drift with another mine does not furnish sufficient ventilation, fans are installed underground. These are multivane blowers, with a capacity of 40,000 cu. ft. per min. against 3 in. water-gage pressure, operated by 50-hp. 2200-volt a.c. motors. Where such installation has been made, the fan is placed on the bottom level, the cage compartment being used as the intake and the skip compartments as the outlet on the upper level, the lathing between these compartments having been made tight. By means of doors on the various levels, the air is forced through all the working places and discharged into the skip compartment on the top level. In order not to interfere with haulage, these doors are opened and closed by pneumatic cylinders controlled, from a distance, by ropes, red and green lights indicating the position of the doors.

Lighting.—In the shaft houses and on the underground plats, 55-volt, a.c. lamps are used; while in the main haulage levels 250-volt d.c. lamps are installed, which obtain electricity from the trolley wire. These are placed at all switches and at intervals from 100 to 200 ft. along the drifts. All men in the mines use carbide lamps.

Telephones.—Telephones are installed at all main level plats, in pump rooms, at underground hoists, in the lander's station in the head frame, in hoisting houses, and in the various surface buildings, such as office and shops. Signaling in the shaft is by means of a.c. electric bells, all mine plats being connected to the engine house on two lines. The repeating bell system is used for operating cages. The cage rider is not permitted to open the door of the cage until the stop bell has been received from the hoisting engineer. A wire-pull bell is always installed in the cage compartment for emergency signals. No method of signaling in the haulage ways is used except that, on large levels operating more than one electric motor, there are colored lights at the entrance to each crosscut, which are automatically lighted when the motor is in that crosscut.

Disposition of Ore after Reaching Surface

During the shipping season, which is from May 1 to Dec. 1, ore from the skips goes directly into standard railway cars and is hauled to the ore docks at Marquette or Escanaba, for shipment by boat to lower lake ports. During the winter months, ore is stock piled from trestles about 40 ft. high, usually built with wooden bents. As wooden trestles must be torn down each shipping season and re-erected in the fall, permanent steel stocking trestles are used at mines of a long life where there is sufficient ore to warrant the larger initial expenditure.

Safety and Welfare Work

Most of the large mining companies employ safety inspectors, who make regular or periodical trips through the mines. A book of safety rules is given to each employee, who signs a receipt for it. Examinations are held on these rules by a special committee. Failure to pass is sufficient cause for dismissal. Once a year, a special committee of workmen visits other mines to compare the safety appliances with rules as enforced in the different properties. This committe makes a number of safety suggestions, which are considered by the officials and almost invariably are accepted. At regular intervals, a certain number of men in each mine are trained in first-aid and apparatus work. Once a year, teams are selected from the various mines and field meets are held to compete for prizes in first-aid work.

The amount of welfare work varies with the different conditions. Some of the companies have hospitals, attached to which is a staff of doctors. In these hospitals, not only the cases resulting from mine accidents are treated, but also cases of sickness in the community. Nurses are employed, who visit the houses of the workmen, give help, and advise in case of sickness. A small uniform charge is paid monthly by all employees to cover doctors' fees for ordinary consultation and visits, while the amount paid by the company for compensation takes care of cases of accidents. Some companies also have a system of pensions. Most of the companies take an active interest in the community by providing in the mine locations, where there are no theaters and other means of recreation, club houses with reading rooms, gymnasiums, bowling alleys, moving-picture machines, etc.

Top-Slicing in Old Fills at El Bordo Mine, Mexico

By R. J. Mechin,* Pachuca, Hidalgo, Mexico

(New York Meeting, February, 1926)

Top-slicing was introduced in the Pachuca district in 1917 by T. C. Baker, at that time mine superintendent of the Santa Gertrudis mine. There then existed 1200 ft. below the surface, lying between the fourteenth and eighteenth levels (a vertical distance of 354.2 ft.) a block of ore having a length of 295 ft. and an average width of 82 ft. and dipping at an angle of 65°.

This block lay in the hanging wall of the vein and consisted of highly altered and fractured andesite, cut through with mineralized quartz stringers. On the foot-wall side, 26 ft. of old filling, left by former operators, was of payable grade. Stoping therein was first undertaken by the square-set filling system, but because of the extremely heavy hanging wall and loose dry filling in the foot wall, all square setting had to be accompanied by spiling, which made the operation exceedingly slow and costly.

A system of mining was sought that would eliminate the difficulties inherent to the square-set system in heavy ground and that at the same time would effect the desired economies in mining. After several years, top-slicing has met all the conditions most satisfactorily, so that when similar conditions presented themselves in El Bordo mine, this method was adopted with necessary modifications.

El Bordo mine is operated by the Santa Gertrudis company, and is situated about two miles north of Pachuca. Because of the diversity of the orebodies, a number of different mining methods are in use, among them top-slicing.

Description of Block of Ore

Former operators, when mining the original vein, left in the foot wall ore of too low grade to be profitable at the time; also, considerabe low-grade material was sorted out and stored as filling in the stopes. Some pillars were left in place, either for support or because they were not payable. All this has formed a body of ore that can now be exploited

* Formerly mine superintendent at El Bordo.

profitably. Its length is 295 ft. and its width varies from 13 to 36 ft., averaging 23.9 ft.

The ore is divided into three classes as follows: (1) "Solid" or ore in place; this is all on the foot wall, or south side, of the vein and requires drilling and blasting; (2) "Compact," probably formed by pillars left in the old stopes and very compact caved material; some parts of this ore require drilling and blasting while the part that can be broken by picking stands well in a 12-ft. slice at about an 85° angle; (3) "Old filling," which is largely mineralized material, caved off the hanging wall, and vein material sorted and left in the stopes during previous mining operations; it is comparatively loose and will not stand for any length of time above an angle of 60°. These three classes occur generally in this order from foot wall to hanging wall—that is, from south to north. The strike of the orebody is approximately east and west, and the average dip 85° north.

The foot wall is hard, unaltered andesite and the hanging wall, where in place, varies from a smooth fault plane to hard slightly altered andesite. Over the larger part of the block being mined the hanging wall is not exposed in place, as unpayable old filling determines the limits of stoping.

Preparatory Work

The preparatory work for top-slicing this block of ore was divided into four parts:

1. Counter-drifts, 5 by 7 ft., were driven on the 305-meter level, the top of the block, and on the 365-meter level, the bottom of the block.

2. Five raises, 4 by 7 ft., were put up in the foot wall at an average distance of 16 ft. from the vein. These raises were spaced at 65.6 ft. centers, and that at the eastern extremity of the series was started 32.8 ft. from the eastern limit of the block.

3. Crosscuts, 3 by 5 ft., were driven off these raises and continued through the vein to the hanging wall in place. Three crosscuts were driven off each raise at 48-ft. intervals, starting from the upper counter drive. This roughly divided the height of the block into thirds and, by using a multiple of 12 ft. these same crosscuts serve as the preparatory work for the fourth, eighth, and twelfth top-slice floors. This exploration work is necessary as there is a tendency for all caved portions of the body of ore to contain holes, which would be a source of danger after stoping had started.

4. The sill floor on the 305-meter level was taken out with square sets, using spiling where necessary. Sills were laid 3 ft. 6 in. in the clear, and a heavy flooring was put down and covered with old timber. The part that had not already crushed was then caved by blasting the stope posts.

Before actual stoping was started, timber stations, 13.2 by 6.5 ft., were cut out on the 305-meter level in front of each raise from the level below. These were designed to hold one day's supply of timber for each stope and are kept stocked from the main timber station near the

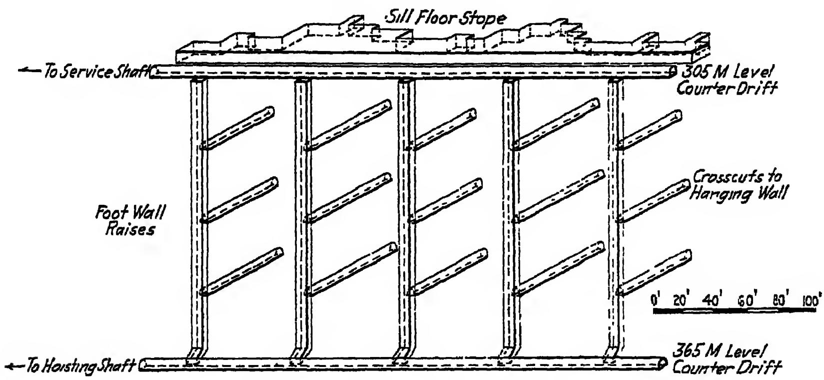

FIG. 1.—PREPARATION WORK.

service shaft. Permanent chute sets were put in at each raise on the 365-meter level. A hand-operated steel gate is used, adopted with slight modifications, from that used by the United Verde Copper Co. at Jerome, Ariz. The preparatory work is shown in Fig. 1.

STOPING OPERATIONS

The work of taking out a complete slice is divided into four parts; the sections numbered 1, 2, 3, and 4 correspond to the numbers in Fig. 2.

1. Preparation crosscuts north, 3 by 5 ft., are driven from the raises in the foot wall until the rock classed as compact is reached.

2. From these preparation crosscuts, drifts are started east and west in the compact as soon as the slice above has been caved, and are continued until all drifts are connected. These drifts are mined 7 ft. 4 in. wide and the full height of the slice, 12 ft.

The solid on the foot wall is taken as the south side of these drifts. The posts are placed in straight lines parallel to the strike of the vein and are offset, when it becomes necessary, through the irregular surface of the solid.

Round posts, 10 in. or more in diameter, by 11 ft. long are used, spaced 4 ft. in the clear from north to south. On the top of the post is placed a cap, which receives the sill of the floor above. This cap is 10 by 10 in. hewn timber, cut in the form shown in Fig. 3. The cap is placed with its long side in contact with the sill of the floor above and long axis parallel thereto; that is, north and south.

In case there is a tendency for the material on the north side of these drifts to loosen, split lagging is put in behind the north line of posts.

This lagging is recovered in the course of mining and is used again in the slice below.

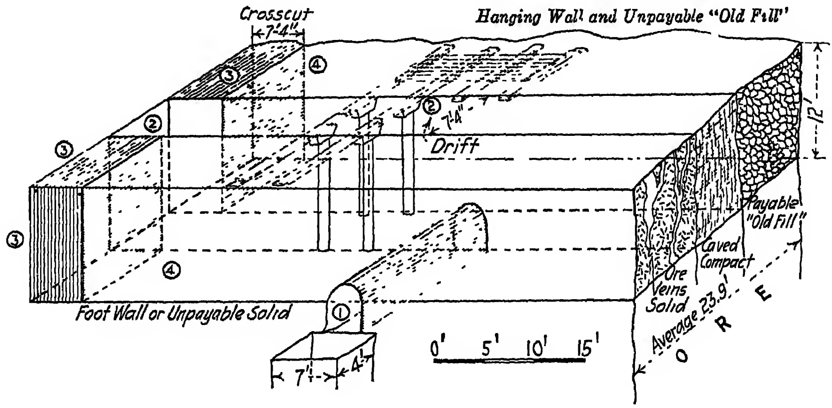

Fig. 2.—Top slicing, el Bordo mine.

3. When two of these drifts are connected, a crosscut north in the compact or old filling and a crosscut south in the solid are started at a point half way between raises. These crosscuts are worked in exactly

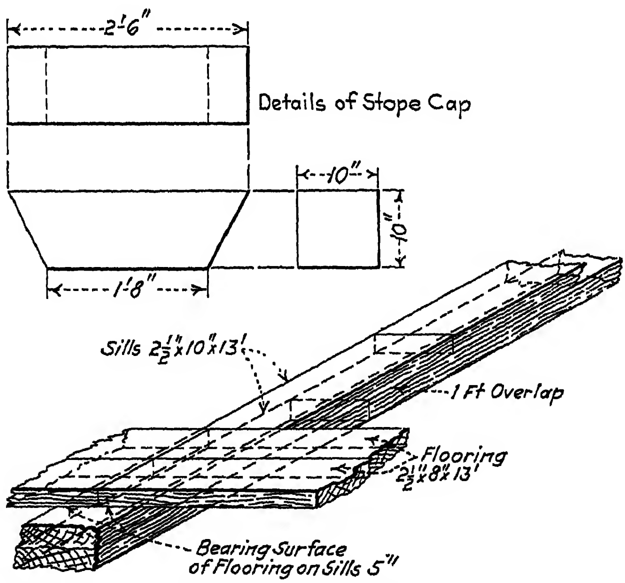

Fig. 3.—Floor and stope cap.

the same manner as the drifts, except that the posts are 6 to 8 in. in diameter by 11 ft. long. The crosscuts to the north are driven to the hanging wall, or to a point where the old filling is unpayable; those to the south to the foot wall or to a point where the solid is unpayable.

4. When these crosscuts are completed, a face the full width of the stope is mined, retreating toward the raises and following closely the metal content on each wall of the stope. The floor is placed as close to the working face as possible to avoid spiling or reinforcing the timber, in case the light posts do not stand the pressure from above. So far, no trouble has been experienced due to failure of the posts before the floor is down. The 10-in. posts in the drifts have proved sufficiently strong to keep these drifts in good condition until extraction is complete and the stope ready to be caved by blasting the posts.

Timbering

The posts in this part of the work are put in 4 ft. in the clear, from north to south, the caps receiving the sills of the floor above, as described in section 2. The space between posts from east to west varies slightly because of the different diameters of posts used.

The sills are of plank 2½ in. by 10 in by 13 ft. and are laid double with a 1 ft. overlap where they join. These sills are placed 3½ ft. in the clear from east to west. The floor is of plank, 2½ in. by 8 in. by 13 ft., which allows a bearing surface of 5 in. on the first, fourth, seventh, etc., sills (see Figs. 2 and 3). A single floor only is put down, care being taken to insure a snug fit around all posts. The outside planks of the floor are nailed to the sills at each end of the plank.

Before caving a slice, all posts possible are recovered for use in the next slice; this varies from 5 to 8 per cent. of all posts put in. Posts are sent down from the framing shop with a 1¼-in. hole drilled at about 20° from the horizontal, when the post is in place. These holes are from 5 to 8 in. deep depending on the diameter of the posts. From one half to a full stick of dynamite is used in about 80 per cent. of the posts still standing when a slice is ready to be caved.

All shoveling is done off flat sheets using round point shovels. Ore is handled in the stopes with wheelbarrows and planks are laid to facilitate tramming.

Drilling

Drilling is done with stoper and jackhammer type of machines of various makes, using 1-in. quarter-octagon and ⅞-in. hexagonal steel, respectively. As soon as two drifts (see section 2 under Stoping Operations) from adjoining raises are connected, a preparation crosscut is started 12 ft. below the slice being mined, from one of two adjacent raises, and the ore from both stopes is thrown down the other raise. In the raise from which the new crosscut is to be driven, a timber compartment and manway are formed of light round stulls and split lagging.

This crosscut is advanced sufficiently before ore from the stope is thrown down what is to be the timber compartment, so that the men

TABLE 1.—*Summary of Costs in United States Currency*

	Ore Mined, Short Tons	Men Employed		Cost of Explosives		Labor Cost		Timber Used, Ft. B. M.				Timber Cost		Lighting Cost		Advance, Linear, Feet	Total Cost	Cost per Ton	Cost per Foot of Advance
								Square		Round									
		Total	Tons Per Man	Total	Per Ton	Total	Per Ton	Total	Per Ton	Total	Per Ton	Total	Per Ton	Total	Per Ton				
Stope preparation	313	777	0.40	$ 505.10	$1.61	$ 675.62	$2.16	3,922.3	12.5	4,310.0	13.8	$ 171.24	$0.55	$ 21.46	$0.07	512.5	$ 1,373.42	$4 39	$2 68
Actual stoping	30,710	9,093	3.38	1,696.49	0.06	13,291.24	0.43	216,196.5	7.0	135,347.0	4.4	7,587.53	0.25	242.31	0 01		22,817.57	0 74	
	31,023	9,870	3.14	$2,201.59	$0.07	$13,966.86	$0.45	220,118.8	7.1	139,657.0	4.5	$7,758.77	$0.25	$263.77	$0.01		$24,190.99	$0.78	

NOTE.—Stope preparation refers to section 1, under "Stoping Operations." Actual stoping refers to sections 2, 3, and 4, under "Stoping Operations."

working in the heading are in no danger. All preparation crosscuts are completed before the slice being mined is ready to be caved. The foot-wall raises are without timber below these crosscuts.

Table 1 gives a summary of the costs and Table 2 the consumption of dynamite.

TABLE 2.—*Consumption and Cost of Dynamite, United States Currency**

	Total Cost	Amount Used, Pounds	Advance per Pound of Dynamite Used, Feet	Cost per Foot Advance	Ore Broken per Pound of Dynamite Used, Short Tons	Cost per Ton
Stoping preparation .	$ 400.30	1765.0	0.290	$0.781		
Actual stoping.......	1219.70	5377.9			5.71	$0.040
	$1620.00	7142.9			4.35	$0.052

* Dynamite used, 1 in. 40 per cent.

Open Stope

Briefly, an open stope is one in which the ore is taken out and no filling is put in; the only support for the walls may be posts or pillars of ore. Such a method is limited to orebodies with strong walls and ore strong enough to stand unsupported over the back of the stope. Details vary and depend on the dip of the vein, mode of breaking ground, method of handling broken ore, and so forth. The examples given are the Beatson, Birmingham district, Mineville district, and Cornucopia.

Geology and Mining Methods of Beatson Mine

BY STEPHEN BIRCH, NEW YORK, N. Y.

(New York Meeting, February, 1924)

LATOUCHE, or the Beatson plant of the Kennecott Copper Corpn., is located in the Prince William Sound district of Alaska about 80 miles west of Cordova and 60 miles from Seward.

Ore was discovered and claims located on Latouche Island in July, 1897. The mines were worked in a desultory manner until 1910, when the property was acquired by the Kennecott Copper interests. The first shipment of ore was made in 1904; only the higher grade ore was mined and no attempt was made to treat the ore until it was taken over by the Kennecott company. Since that time the mine has been developed to produce 1500 tons a day and a mill has been erected for treating this tonnage, using flotation entirely. The orebody is more or less lenticular in shape with a maximum width of about 280 ft. and a length of about 800 ft. The southern end of the lens is split by a horse of waste for about 400 ft. The hanging-wall limit is a well-defined fault associated with a band of pyrrhotite, having an average dip of 60°. There is no defined foot wall, the value of the ore governing the limits of the mining; in one part of the mine, however, it is defined by a minor fault. The orebody is in a shear zone of the country rock of graywacke and slate. The principal mineral is chalcopyrite associated with pyrite and quartz.

The mine is situated a few hundred feet from tidewater; the mill is on the beach and the ore is hoisted direct from the mine through a vertical shaft into the mill bins. Until the past year practically all the mining had been conducted in open pits. During the past two years, the ore above the 200-ft. level has been developed and a system of stoping devised to recover this ore; however, this system has not been in use long enough to give any definite results as to costs or efficiency. It consists, principally, of dividing the orebody into stopes, across the full width of the orebody, 70 ft. wide with a pillar 30 ft. wide between the stopes. Raises are driven through the stopes at intervals of about 60 ft. and all the ore is broken by drilling from these raises; in other words, similar to a shrinkage stope, doing the drilling from the raises instead of setting up on the broken ore as is customary. The ground is very much broken up by clay slips and seams running in every direction. Shrinkage stoping had been

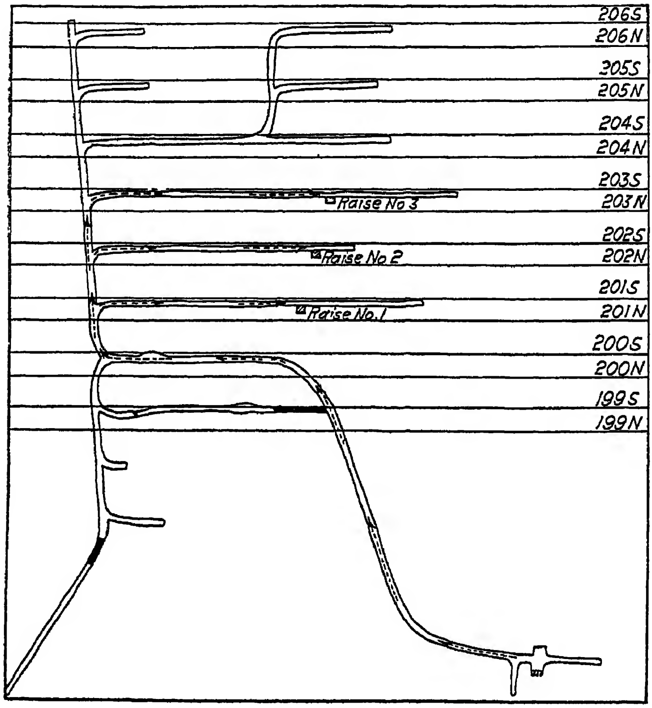

FIG. 1.—THE 200-FT. LEVEL.

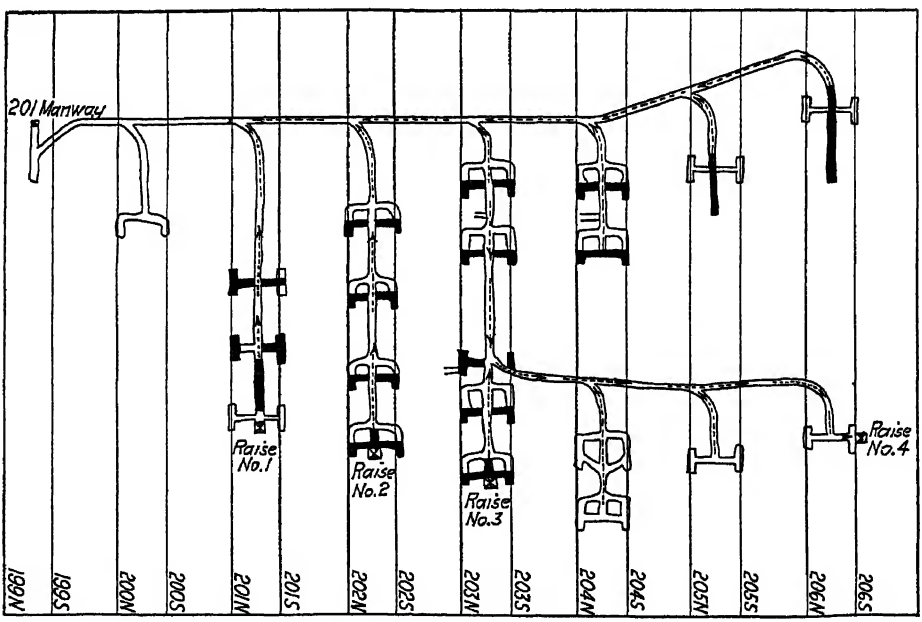

FIG. 2.—THE 150-FT. LEVEL.

unsuccessful because of the time necessary to bar down and the impossibility of making the back safe, because of these slips. On the upper level, in the higher grade ore, some square setting was done, but the average grade of the ore precludes the use of this system throughout the mine.

Some exploration work has been done with the diamond drill, but for the greater part drifts and crosscuts have been driven to explore the ground. In sampling, all crosscuts and drifts, as well as raises, are sampled in 5-ft. intervals and a double groove 1½ in. deep by 6 in. wide is cut in each working place. It has been found that results, even with such large samples and double sampling, are approximately 20 per cent. too high. It is estimated that 12 cu. ft. in place and 20 cu. ft. broken produce a ton of ore.

Underground Mines

Mine Openings, Shafts, or Tunnels

The mine has two entrances, a vertical shaft and a main-level tunnel. The main-level tunnel is 970 ft. long from the portal to the central point of the orebody. It is about 7 by 7 ft. in size. Where timber is used, the measurements are 6 by 6½ ft. inside the timber. The shaft is 360 ft. deep with three compartments, each compartment being 5 by 5 ft. inside the timbers. Loading pockets are located below the main level and below the 200-ft. level.

Underground Development Plans

The 200-ft., 150-ft., and main levels are shown in Figs. 1–3. The curves on the 200-ft. level have a radius of 35 ft.; on the 150-ft. level a radius of 30 ft. is used. All the main haulage tracks have 35-lb. rails; on the 150-ft. level, 16-lb. rails are laid, this track being used only for hauling supplies. Track grade is carried uniformly at 0.5 per cent.

The main compressed-air line is 6 in. No. 20 gage galvanized ventilating pipe, 10 in. in diameter, is used.

Mining

Drilling and Blasting.—The following is a list of drills used, together with the size of the steel:

	Shape and Size of Steel
Ingersoll-Rand, BCRW-430 Jackhamer	⅞-in. hollow hexagon
Ingersoll-Rand, CC-11 stoper	1⅛-in. cruciform and 1-in. quarter octagon
Ingersoll-Rand, 248 Leyner	1¼-in. round hollow
Ingersoll-Rand, BCR-430 Jackhamer	⅞-in. hollow hexagon

The single-taper cross bit is used on all steel. Holes are tamped with mud enclosed in parafine paper cartridges, and firing is done with fuse and caps or with electric blasting caps and magneto.

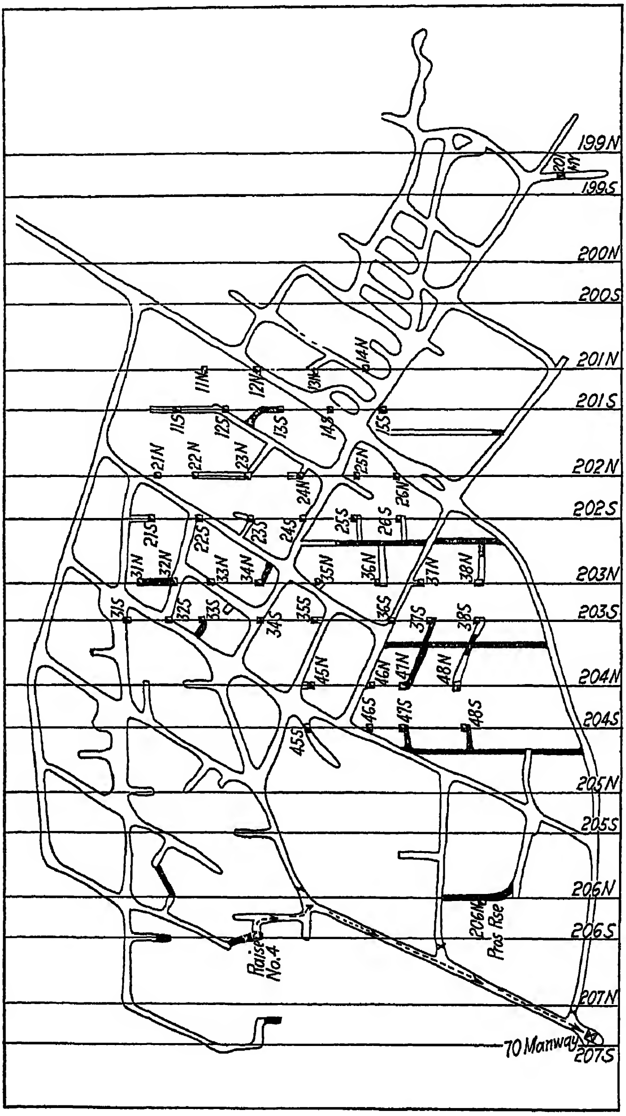

Fig. 3.—The Main Level.

The explosives used are 40 per cent. gelatine and 40 per cent. Red Cross manufactured by the du Pont company.

Tramming and Haulage.—Granby 3 and 4-ton side-dump cars are used, dumping direct into the shaft ore pockets.

Baldwin-Westinghouse storage-battery 4½-ton locomotives furnish the motive power. The rails are 35 lb., the gage 24½ in., and the grade 0.5 per cent.

Underground Storage and Dumping.—The underground storage consists of the two shaft ore pockets, each with a capacity of about 250 tons. The cars dump directly into these pockets, the ore then being drawn to the measuring pocket in the shaft and then loaded into the skips.

Hoisting.—The man hoist is a Coeur d'Alene Hardware & Foundry Co , electric hoist having a 36 by 36-in. drum driven by a 100-hp., 440-volt

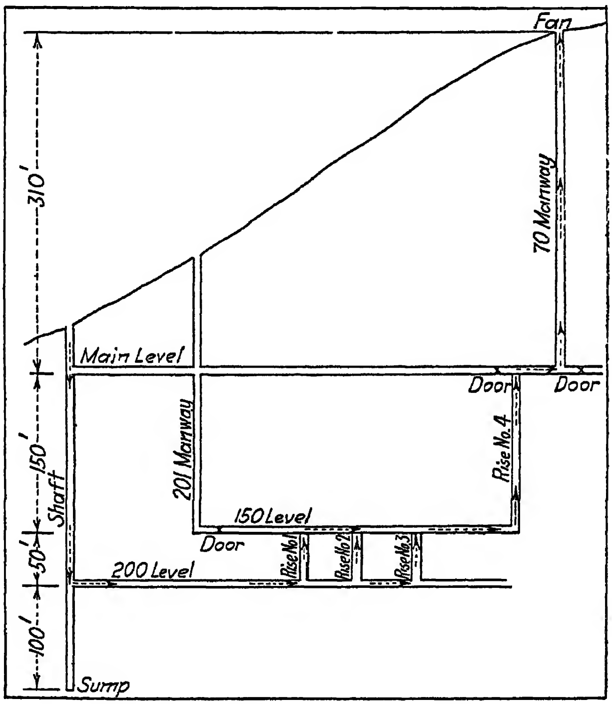

FIG. 4.—DIAGRAMMATIC SECTION SHOWING VENTILATION RAISES.

motor. A single-deck cage with spring safety dogs and ¾-in. plow-steel rope is used.

The ore hoist is a Wellman-Seaver-Morgan double-drum geared electric hoist having 60 by 42-in. drums driven by a 150-hp., 440-volt Westinghouse motor.

The cable used is 1⅛-in. plow steel; the skips hold 4 tons and are operated in balance; the hoisting speed is 600 ft. per min.

Pumping.—The pumping is not a serious problem; the following electrically driven pumps take care of the entire mine: One Gould triplex pump, 70 gal. capacity; one Gould triplex pump, 275 gal. capacity; one Gould triplex pump, 80 gal. capacity.

Air Compression.—One Chicago Pneumatic Tool Co. compound compressor of 1730 cu. ft. capacity driven by a direct-connected, 2200-volt, 295-hp., synchronous motor supplies the mine with compressed air.

Ventilation.—Natural ventilation has been relied on but with increase in the amount of bulldozing on the 150-ft. level, attendant with increased mining underground, an electrically driven fan of 30,000 cu. ft. capacity is being installed. The ventilation system is shown by the dotted lines and arrows on the accompanying figures.

Lighting.—The mine is lighted by electricity, the circuit being 110 volts. Several types of globes have been used—carbon, mazda type B, and mazda mill type. The mazda mill-type 50-watt globe has been found the most satisfactory. Small carbide cap lamps are used by the miners.

Telephone.—The telephone is used for general underground communication. Electric signals are used in the shaft.

Records of Unit Production

For the year 1922, the total production was 274,863 dry tons (2000 lb.) ore milled. Tonnage broken amounted to 349,071 tons. The following data are based on the tons broken and not on the tons milled.

Work in the mine is done either on day's pay or on the bonus system. Bonus is paid on footage driven, in development work, or on footage drilled, in stoping.

Bluff mining, 92,365 tons broken.	
Tons per man per hour	9.68
Man-hours per ton	0.103
Stopes, 226,199 tons broken.	
Tons per man per hour	9.30
Man-hours per ton	0.107
Stope preparation and development in ore, 30,507 tons broken.	
Tons per man per hour	0.661
Man-hours per ton	1.51
No work done in rock or waste.	
All underground labor:	
Tons per man per hour	1.76
Man-hours per ton	0.57
Surface force:	
Negligible.	
All mine labor:	
Tons per man per hour	1.67
Man-hours per ton	0.60

Classification of labor:

	Per Cent.
Foreman and bosses	4.1
Miners	43.8
Helpers	12.3
Muckers	4.1
Trammers and loaders	8.2
Timbermen and helpers	5 5
Hoistmen and skip tenders	9.7
Miscellaneous	12.3

Total labor turnover for the year 1922 for the entire plant was 248 per cent.; for the mine alone, 235 per cent.

Total labor cost, expressed in percentage of total cost of mining, was 59.1 per cent.

Records of Units of Supplies Used

Explosives used on ore-breaking on bluff ore was 0.3 lb. per ton; on stoping, it amounted to 0.3 lb. per ton broken.

All timber, lagging, and poles used amounted to 0.020 ft. B.M. per ton ore broken.

Power required was:

	Horsepower-hour per Ton
Mining (compressed air)	1.52
Haulage and hoisting	1.10
Pumping	0.38
Ventilation	0.24
Lighting	0.16
Total	3.40

(For discussion, mainly on the Kennecott mine by the same author, see page 510.)

Mining Methods at Cornucopia, Oregon

By Robert M. Betts, Cornucopia, Ore.

(New York Meeting, February, 1925)

The Cornucopia district is situated in northeastern Oregon 25 miles from the Snake River branch of the Oregon Short Line R.R. and 75 miles northeast of Baker City, in the Wallowa Mountains.

The formation is principally granodiorite, which has intruded much older rocks locally known as greenstone, and which are probably altered schists. In the upper horizons of the veins, greenstone is found as inclusions in the granitic mass. Both of these formations have been cut by numerous basalt, aplite, and porphyry dikes.

Veins of the district are the result of faulting and the consequent crushing of the wall rock. The ore has been deposited irregularly along these faults and formed a lenticular type of vein. The ore varies in width from a few inches to 20 ft.; the average stoping width is about 5 ft.

As the basaltic dikes are subsequent to the ore deposition, they are at times a hindrance in development and mining operations; on the other hand, a dike which has followed a vein as a line of least resistance often serves as a good wall in place of the crushed, rather loose granite. The veins generally strike northeast-southwest and dip about 35° to the west. Orebodies are from 100 to 300 ft. in length and about the same in height. The lenses are irregular; inclusions of wall rock are common in the ore and are a factor in reducing the grade.

The ores are quartz, carrying some free gold near the surface, but the metals for the most part are in iron sulfides, sometimes associated with chalcopyrite. The gold and silver are in a mechanical combination with the iron, which requires find grinding to liberate them. The ratio of silver to gold is 5 to 1, by weight.

Development

The Cornucopia Mines Co. owns, and operated until 1920, two mines and two mills of 100 tons capacity each. The sustained high costs and labor shortage, however, necessitated the abandoning of the Union mine, the oldest in the district. Two crosscut tunnels were then started, first, to cut the Last Chance vein at a much lower horizon and, second, to prospect two parallel veins. The ore from the Last Chance vein will go to the old Union mill and the ore from the new veins to the Last Chance mill.

The Last Chance crosscut, 4700 ft. in length, has been completed and a 1250-ft. raise driven to connect with the lower workings of the mine. Development work is in progress but full-time milling has not been resumed and the second crosscut, known as the Lawrence tunnel, has not reached its objective.

Mining

Stopes are stulled and no attempt is made to fill them. Drifts and crosscuts rarely have to be timbered, but some trouble is experienced in holding the hanging wall, which slabs off and dilutes the grade of ore sent to the mill. It is roughly estimated that 8 per cent. of the tonnage is composed of this waste. At times, some hand sorting is resorted to in the stopes but, as a rule, the fragments are too small to be discarded. Horses of waste occur in the vein and usually must be taken, but whenever possible they are left as pillars.

Hand sampling of the faces is not attempted, but samples are taken from chutes and from broken rock in the drifts after blasting.

Tramming on main levels is by storage-battery locomotives, and on other levels by hand with 16-cu. ft. cars.

Costs

As the extraction of ore was of secondary importance, compared to the development work, costs per ton for the full year of 1923 would be misleading; but in the three months period when full tonnage (9014 tons) was run in the Union mill on Last Chance ore the costs were as follows:

	Per Ton
Mining	$1.94
Milling	1.85
Surface	0.25
Marketing	0.07
Overhead	0.51
Total	$4.62

The cost of development work for the period was 86 cents per ton.

Unit Production

A summary of the record of unit production is as follows:

	Tons per Man-hour	Man-hours per Ton	Per Cent.
Miners in stopes	2.118	0.472	
Miners developing ore	1.663	0.601	
Miners developing waste	1.642	0.609	
All miners	0.1994	0.501	
All underground labor	0.4256	2.349	
Labor turnover	......	.....	30
Total labor cost to total cost of mining	......	.....	53
Contract labor	......	.....	23
Day labor	......	.....	77

Consumption of Supplies

Based on 9014 tons mined, the following supplies were used:

	Quantity	Per Ton
Explosives	8,650 lb.	0 96
Fuse	26,332 ft.	2.92
Caps	4,300	0.48
Timber	2,568 lin. ft.	0.28
Power:	2 hp.	
Compressed air		1.5
Haulage		0.25
Hoisting		0.25

Red Iron Ore Mining Methods in the Birmingham District*

By W. R. Crane, † Birmingham, Ala.

(Birmingham Meeting, October, 1924)

Mining of the red iron ores of the Birmingham district has been carried on energetically during the past 50 years, and their development has created a large iron and steel manufacturing center, the only important one in the South. The district produces approximately 10 per cent. of all the iron ore of the United States (80 per cent. of the Alabama ore mined is red ore); also, 40 of the 400 blast furnaces of the country are in the district tributary to Birmingham. The rapid growth of the district has been made possible through investigations that resulted in radical changes in furnace practice, and a still greater impetus will come from the study of the low-grade, high-silica ores, as a result of which they will be made amenable to treatment by concentration.

Mining practice in this district has been comparatively simple because of the occurrence of the ore; but with the rapid extension of the workings and the disturbed condition of the ore bed at some distance from the outcrop, more difficult conditions are encountered and the tendency is toward worse rather than better conditions.

Should the high-silica ores of the lower bench of the Big Seam become available through beneficiation, mining practice will have to be modified to meet the new conditions, which will be rendered more difficult by the increased weight of cover that will exist at considerable distances from the outcrop. Support of workings will require greater attention and the efficient and economical operation of the mines will depend largely on the successful solution of the problems of working and handling the ore.

History and Early Development

The first explorers of the coal and mineral lands of Alabama were blacksmiths and mechanics mustered out of the army after the war of 1812. These men recognized the red rock of the Birmingham district as iron ore, and utilized it in making cooking utensils and farm implements. The first blast furnace was built and operated at Russellville in 1818, where also was a foundry and rolling mill. Soft ore was used at that time, but, in 1864, the first red hematite ore of Red Mountain was smelted, near Irondale. Birmingham was founded in 1871, and the successful use of coke in making pig iron in 1876 was the beginning of

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great industrial activity in the district, which now contributes 7 per cent. of the pig-iron production of the United States.

Prior to the use of the hard ores, the soft ores were mined by open-cut methods. These surface workings follow the outcrop of the Big Seam for 25 miles along the crest of Red Mountain, extending in many places down the dip of the orebed for several hundred feet, or until the cover reached a thickness that was prohibitive to move, even with the cheap labor of those days. The bed was stripped by hand, the economic limit of thickness of cover being 28 ft. Mining the exposed ore was then comparatively simple, it being blasted and quarried. Mining contracts were let at that period for as low as 35 cents a ton.

Only 3 ft. of the ore of the Big Seam was worked at first; this was increased later to 6 and 8 ft. As the bed averages from 18 to 20 ft. in thickness, evidently there remains a large tonnage of ore in the open cuts. However, considerable of the lower bench ore left in the open-cuts was mined subsequently by underground workings.

With the curtailment of open-cut work, because of excessive overburden, development by drifts and slopes was begun. Full advantage was taken of the lay of the ground in order that drifts might be used instead of slopes, thus eliminating hoisting and pumping. As a result, practically the whole area adjacent to the outcrop was mined out in a relatively short time.

Before the soft ores were exhausted, the hard ores had been encountered and the problem of their utilization solved, to the extent at least that they were mixed with the soft ores. Development of the hard ores was so rapid that they became the more desirable and the mining of the soft ores was soon abandoned. Mining of the hard ores was continued in the slopes and drifts, but the number of these openings has gradually been reduced until at present there are only 23 slopes in the district.

Geology and Mineralogy of Ore

The hematite ore of Red Mountain, commonly spoken of as "red ore," occurs in the Clinton formation of the Silurian age. This formation varies from 200 to 300 ft. in thickness and is composed largely of shale and sandstone.[1] There are four orebeds in this formation, but only the Big Seam and Irondale are considered here; the former is by far the more important from the standpoint of grade of ore and amount available. They both outcrop on the west face of Red Mountain and dip southeast at approximately 16°. The strike of the orebeds is northeast and southwest, and maintains that direction with remarkable uniformity.

The Big Seam ranges from 15 to 22 ft. in thickness, but as it is separated into two parts by slate from knife edge to 30 in. in thickness, it is

[1] E. F. Burchard and Charles Butts: Iron Ores, Fuels, and Fluxes of the Birming-

not possible to give for the whole bed an average thickness that would be intelligible. The subdivisions of the seam are known as the upper and lower benches, and average about 10 and 8 ft. in thickness, respectively. The Irondale bed ranges from 3 to 8 ft., but averages under 6 ft. where it has been worked. The most important occurrence of ore is on east Red Mountain, between Birmingham on the north and Bessemer on the south. Beyond these limits, the workable deposits extend for considerable distance, but they are thinner and usually of low grade.

The soft, or leached, ores occur at the outcrop and extend down the dip of the beds for distances varying from a few feet to, in extreme cases, 400 or 500 ft., with probably an average depth of between 200 and 300 ft. The contact between the leached and unleached (hard ore) portions of the bed forms a very irregular line. The leaching action has, in large measure, affected the associated overlying formations, rendering them less strong, and complicating the drainage problem by making impervious beds porous.

The orebed is disturbed and warped out of its normal position by the occurrence of both anticlinal and synclinal folds. Secondary folding has also taken place, complicating conditions resulting from normal folding and faulting. Canoe-shaped basins occur, the variations in width and often abrupt termination of which seriously derange the regular development of the mines; not the smallest difficulty is the termination of main slopes, requiring the establishment of auxiliary slopes and haulageways. Wide variations in dip of the orebeds mark the occurrence of hills and basins that have been produced by folding of the strata, which may in itself seriously affect conditions of mining.

The ore-bearing formations are cut by numerous faults, which roughly parallel the outcrop along Red Mountain, although a number of the larger faults cut across the mountain, forming gaps. Most of the faults are normal, but a few thrust or reverse faults occur. Displacements of orebeds range from a few feet to several hundred feet, and require much rock work in continuing the slopes and changing grade of tracks.

Although the grade of the red ore is remarkably constant for any given horizon (it only varies about 2 per cent. in iron for the district) in the upper and lower benches of the Big Seam it varies considerably. The lower bench is higher in silica. The average iron content of the ores mined at present is 35.84 per cent. (say 36 per cent.), while the range in composition is: iron 34.36 to 36.41 per cent.; lime 15.74 to 19.10 per cent.; silica 9.68 to 12.78 per cent.; alumina 2.75 to 3.38 per cent., and phosphorus 0.267 to 0.325 per cent. This ore contains no sulfur (see Fig. 1).

Associated with the orebeds are numerous beds of ferruginous sandstone, which vary greatly in thickness and in iron, lime, and silica contents; in certain localities they are surprisingly high in iron, three samplings giving 21.23, 22.97, and 32.30 per cent., respectively. If a

process is perfected for beneficiating such material, it will add considerably to the reserves of ore.

The average true and apparent specific gravities of the red ore are 3.61 and 3.48 and the average porosity 3.48. The weight is approximately 221 lb. per cu. ft., which gives 10 cu. ft. to the long ton. Freshly broken to settled ore in piles varies from 20 to 17.2 cu. ft. per ton. Preliminary tests to determine the ultimate or crushing strength of the ores show that they will probably sustain with safety 8000 to 10,000 lb. per sq. in. of static load, but will fail under loads of 18,000 to 20,000 lb.

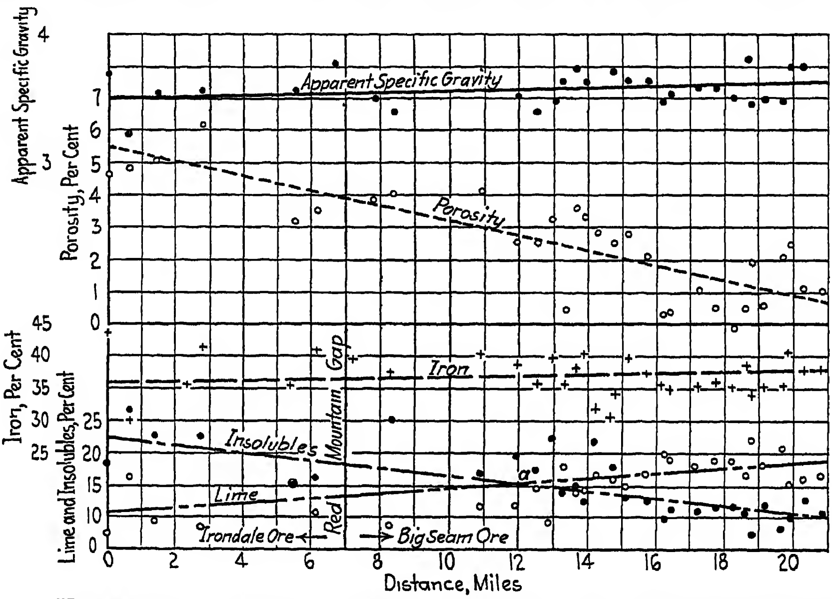

FIG. 1.—VARIATIONS IN SPECIFIC GRAVITY, POROSITY, AND CONSTITUENTS OF ORE.

The ore is sufficiently strong to support considerable thickness of overburden, provided that the cross-section of the pillars regularly conforms to a standard minimum dimension, varying, of course, with conditions.

MINING METHODS EMPLOYED

Working the iron ore of Red Mountain is, in many respects, relatively simple, because the ore occurs in beds of remarkably uniform thickness and dip; this applies equally to development and mining. Underground mining began in 1875; the present rate of production is over 5,000,000 tons annually.

The development methods used in the early days are largely in vogue and there is little likelihood of any material change for many years. Slope operations are thoroughly established; there are only two shafts in the district, but the writer believes that vertical shafts could probably be made to serve to the best possible advantage in developing and working a large area; however, this will be discussed in another place.[2]

The methods of mining the Big Seam are: (1) mining the upper bench only; (2) mining the upper bench at first mining, then the lower bench at second mining; (3) mining the upper and lower benches together; and (4) robbing of pillars. As the upper bench is worked exclusively in most of the mines, the method of development employed under such conditions will be considered first.

Development of Mines

Slopes are driven on the dip of the orebed, beginning with the outcrop if possible, and are spaced at more or less regular intervals along the

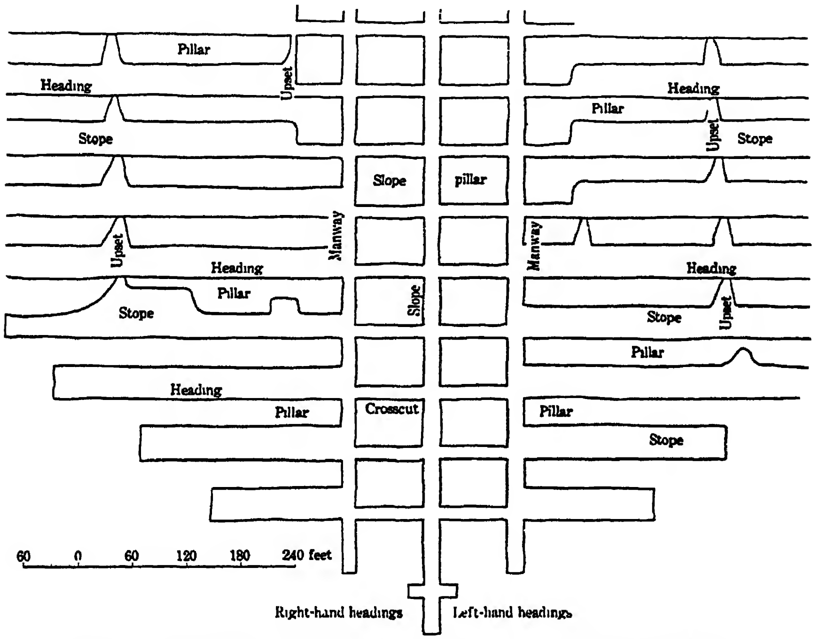

FIG. 2.—GENERAL PLAN OF DEVELOPMENT OF AN IRON-ORE MINE.

outcrop. The average distance between slopes in the various groups of mines is 1333 ft., 1480 ft., 2350 ft., and 2475 ft. Regular dips permit slopes being driven with great regularity, but the occurrence of folds, cross folds, and faults necessitates radical changes being made in the driving of slopes, and results in extreme irregularity of grade. Faults of small displacement do not materially affect the grade of the slopes unless a considerable change in dip occurs below and beyond the fault. But faults of large displacement necessitate radical changes, often requiring much rock work and long stretches of track of steep grade. In brief, the handling of ore is naturally rendered much more difficult than in mines free from faults and folds.

Slopes are usually driven 14 to 16 ft. wide and the full height of the ore, except where the top rock is weak, in which case slopes are 7 to 8 ft. high, leaving 2 to 3 ft. of ore as support.

Paralleling the slopes, and driven 75 ft. on either side of them, are the manways. These are usually driven 12 to 15 ft. wide and as high as the workable orebed. They are used for the ingress and egress of men, sometimes for mules, and serve to a limited extent as ventilating passages. In the larger mines, the men are taken to and from their work in cars and skips, and in several mines the manways have been abandoned.

From either side of the slopes, other passages are driven into the orebed, and are extended at slight grade for several hundred feet laterally. These passages are called headings, and divide the orebed into panels or lifts in which the breaking of ore or stoping is done. Recent practice is to drive the headings 65 ft. apart, but intervals of 80, 100, 120 ft. and upwards indicate a marked change in method, which probably will eventually be established at or close to 200 ft. This change has been brought about by the adoption of such mechanical loading devices as scrapers.

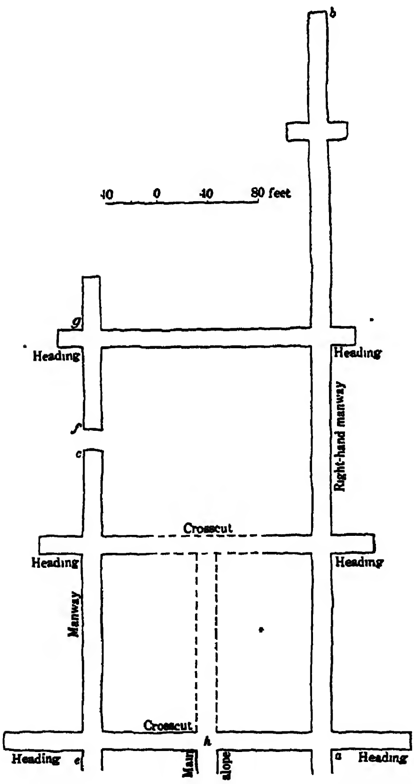

Fig. 3.—Method of advancing manways and slopes.

The headings are driven from the slopes to the manways; they have a width of 15 ft. and a height the full thickness of the orebed worked. Beyond the manways, the headings are increased to a width of 30 or 35 ft., after which they advance with the stopes as one and in a single operation. The general development of a mine is shown in Fig. 2, where the work is being done symmetrically; that is, on both sides of the slope. Extension of a slope and manways, however, cannot be carried symmetrically if proper protection is given the working force, consequently the manways are driven one or two headings in advance of the slope, one manway being considerably ahead of the other. A crosscut is then driven toward the other manway and where the slope would be, and the slope is raised—that is, driven up the dip—to connect with that part of the slope already completed. By this means miners are not exposed to the danger of ore and rock falling down the slopes while hoisting is under way, nor are they in danger from runaway cars or skips (see Fig. 3).

With car or trip haulage in slopes, development is modified somewhat the heading openings along the

slopes. Such an arrangement is necessary to permit of the placing of switches in the slope tracks, otherwise, because of the number of frogs, danger of derailment would be greatly increased.

While car or trip haulage on slopes seems to be a simple and ideal arrangement, eliminating the additional handling of the ore in transit from stope faces to the rock house on the surface, the use of cars in stopes and skips on slopes is much to be preferred, so skip haulage is now used in most of the mines on Red Mountain. This change has resulted in a

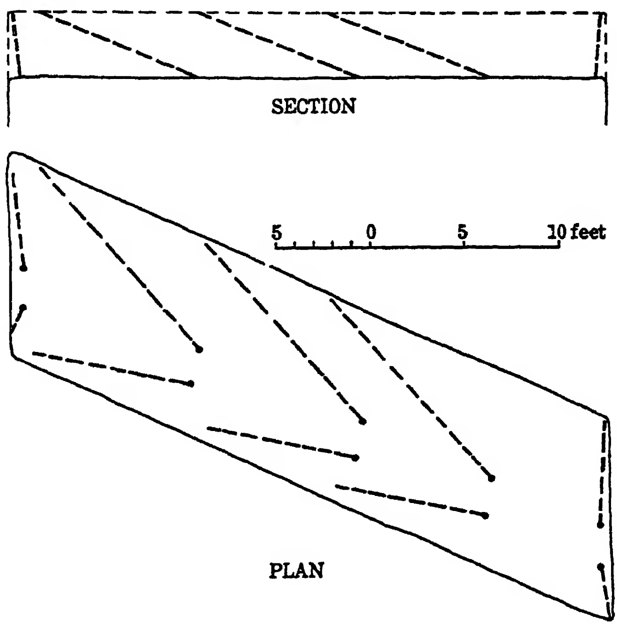

FIG. 4.—FACE OF HEADING AND STOPE, SHOWING ARRANGEMENT OF HOLES.

change of development, mainly the driving of slopes in the foot wall or formation underlying the portion of the orebed worked, and the turning off of headings at equal distance and opposite each other along the slopes. Loading stations are built along the slopes, one for two opposite headings, thus reducing the amount of development required and facilitating the handling of ore.

Driving Headings and Stopes

The driving of slopes, manways, and headings constitutes development in these mines, while stoping and forming upsets and break-throughs may be considered as mining; at the same time, as headings and stopes are nearly always driven as one operation, they will be described as one.

The height of headings and stopes is usually that of the workable bed of ore, although occasionally roof ore is left to support the top rock or draw slate. The roof ore may be, and often is, mined in a subsequent

operation, but such work does not constitute a part of the original stoping operation. The height of stopes ranges from 9 to 14 ft., and averages 10 ft. for all the mines. The width of stopes varies from 25 to 45 ft., but 35 ft. is most common. The combined heading and stoping face, which is 10 ft. high and 35 ft. wide, is carried foward in one operation, the method of attack varying mainly with dip of orebed.

The general arrangement of holes is shown in Fig. 4, but in this work advantage is taken of the condition of the working face; that is, shape of face and number and direction of slip planes, the latter having the most influence.

It requires two 8-hr. shifts to complete a round of holes, and consequently, one full advance of the heading and stope face; but to maintain a continuous supply of ore from each working face, blasting is done at the end of each shift. From 50 to 65 tons of ore is broken per shift, or, for a full advance of face to a depth of 4 ft., 120 to 130 tons are produced.

Mining the Upper Bench

A mine that has been developed according to usual practice for hand work (that is, mucking including handling and shoveling) has the head-

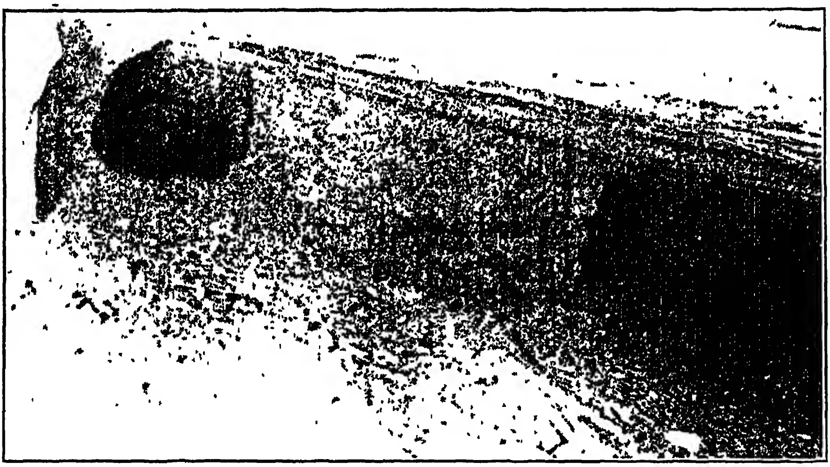

Fig. 5.—Arch pillars cut by break-throughs.

ings driven at intervals of 65 ft., so the combined headings and stopes have a width of 35 ft., which leaves an arch pillar 30 ft. wide between stopes. At intervals of 150 to 200 ft. along the stopes, upsets are driven through the pillars; in most cases they break through the pillars forming the so-called "break-throughs" (see Fig. 5). The primary object of the break-through is to provide outlets for air currents by which dust and fumes from explosives may be promptly removed from the working places; also to permit the passage of men from heading to heading and, where it is necessary or desirable, to run air or water lines from one heading to

another. An additional function of the upsets is to remove as much ore as possible in the preliminary operation of mining the ore.

As shown in Fig. 2, upsets are irregular in shape and become narrower as they are advanced to the next heading. The arrangement of holes commonly used, shown in Fig. 6, is similar to that used in stoping. The driving of upsets is similar to slope work, particularly as both are driven up the dip, but differs in that the full height of the workable orebed is always removed; upsets are similar in practically all respects to raise stopes commonly used in metal mines. The amount of ore broken by a full round of holes in driving upsets is from 90 to 110 tons for each 4 ft. of advance.

Change in Mining the Upper Bench

During the past 10 years, numerous changes have been made in mining practice, particularly in the greater width of stopes and methods

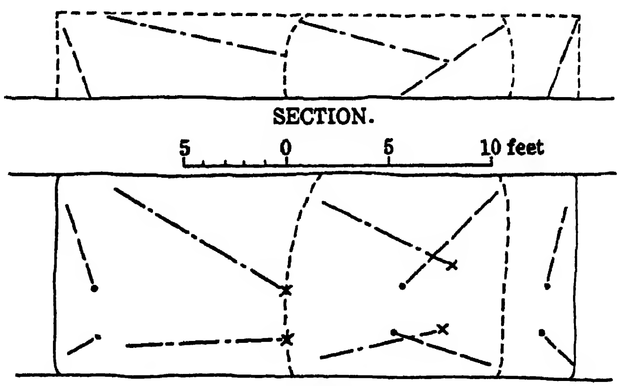

FIG. 6.—UPSET SHOWING ARRANGEMENT OF HOLES.

of attack in mining. The distance between headings has increased from 55 to 200 ft., although the headings and preliminary stopes have not been increased appreciably, in fact, in the case of the 200-ft. interval, 20 to 25 ft. headings are used. The increased interval between headings has resulted in much larger pillars from which the larger part of the ore mined is obtained.

Increasing the size of stopes requires more attention to be given to the support of the top rock, particularly in certain mines in areas where the condition of the top rock is bad. It is recognized that there is a limit to the size of the area that can be worked out, even with such temporary support as can be provided.

In contradistinction to pillar robbing, wide upsets were driven as a secondary operation to the original mining work, which were opened from the preliminary stope and driven directly up the dip of the orebed.

With heading intervals of 70 to 75 ft., headings being 35 ft. wide, the 35 to 40-ft. pillars were cut into blocks 25 to 35 ft. long by upsets

or raise stopes 25 to 30 ft. wide. The ore, as broken in the stopes, was loaded into cars operating on tracks run to the stope faces from the headings.

Later, stopes were driven 35 to 40 ft. wide and from them upsets made 75 to 100 ft. wide; these were spaced from 75 to 100 ft. apart and were carried to the heading above. This work was first done with heading intervals of 75 to 80 ft., the pillars being offset or staggered in the adjacent headings so as to afford maximum protection (see Fig. 7). A wide face of ore in the stopes made it possible to take full advantage of all conditions favorable to drilling and blasting; but in order that the ore might be handled to the best advantage, tracks were laid from the main headings to the raise stopes. In special cases where the dip was

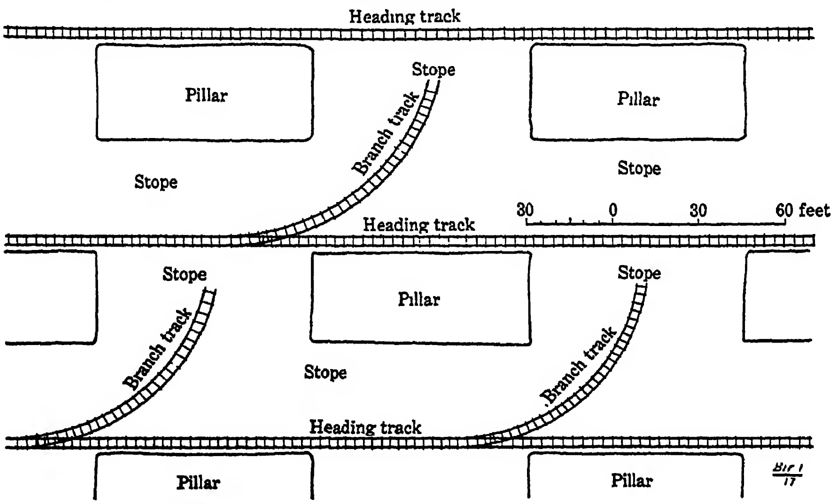

FIG. 7.—USE OF BRANCH TRACKS IN WORKING WIDE STOPES.

moderately high, gravity planes were used. With higher dips and with better conditions of top rock, stopes were driven in line for several adjacent stopes, the ore being drawn therefrom to their respective headings, even through two or more stopes to a heading track that served as a main haulageway to the slope.

Following the more or less successful working of the large stopes, the method was extended to heading intervals of 160 ft. and to fairly high dips, giving pillars 125 ft. wide. The preliminary raise stopes are driven 25 to 30 ft. wide, and entirely through the pillars to the headings above. The ore is handled by gravity planes on the full dip of the ore-bed, as shown in Fig. 8. On completion of the preliminary raise stopes, a new stope is started at the bottom of the pillar and is carried upward, thus removing successive strips or slices of pillar parallel with the dip until the stope is increased to a width of 130 to 150 ft., or to such a point that further increase in size of stope threatens to weaken the roof unduly.

Numerous props are used in the stopes, being placed close together and in rows along the dip, regularity in placing them being required by the gravity planes.

Use of the scraper loader has been responsible for the more extended application of the wide stope method. This device requires a relatively long movement of the dip (best results are on a dip of 15° to 20°) of the orebed, consequently a long heading interval and, for economical operation, a fairly wide stope. The lower limit of stope interval for scraper loaders is probably 150 ft.; the upper limit has not been definitely ascertained, but will vary largely with the particular character

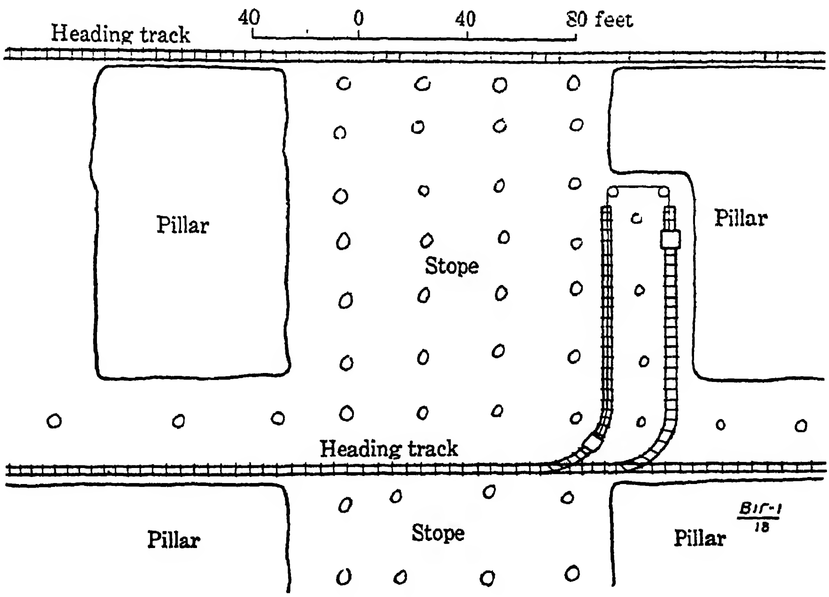

FIG. 8.—USE OF GRAVITY PLANE IN WORKING LARGE STOPES.

of roof. The length of the stopes has, of necessity, been increased to correspond with the increased heading interval; but every effort is being made to determine the safe working limits, as influenced by conditions of top formations, occurrence of ore, and dip of orebed. It is evident, therefore, that while scrapers are reducing the cost of development work, they are introducing into the working of the mines an element of danger in connection with support, which did not exist, except in a limited way. Although pillars are left, these are not altogether desirable where scrapers are operated.

While scraper loaders have been used in the mines for a number of years, it is only comparatively recently that they have reached a state of development in which their successful application is assured. Their most simple application, as shown in Fig. 9, consists in driving headings

18 to 20 ft. wide and 150 ft. apart. At intervals of 75 ft., upsets are driven in the pillars, which are 20 ft. wide and 20 ft. long, beyond which point they are increased in width to 50 ft., with ribs extending up the dip at an angle of approximately 60°. When the full width of the stope has been developed, it is carried up the dip by overhand stoping, the stope face maintaining as nearly as possible and practicable a line parallel with the headings, and ultimately breaks through the pillars throughout practically the full length of the stope.

In Fig. 9, the working face of the stope is shown by the line *ab*, into which the stubbing bar of the drag line is set, and along which it is

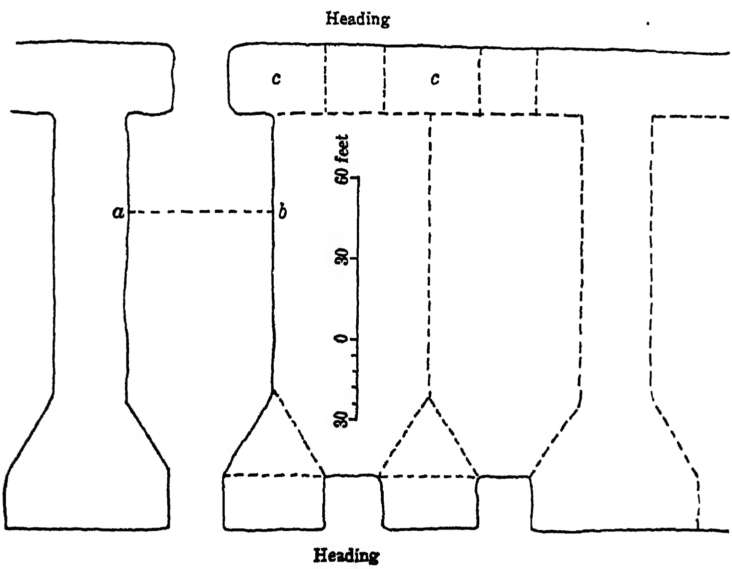

FIG. 9.—ARRANGEMENT OF STOPES FOR SCRAPER LOADERS.

moved as the operation of the scraper is shifted from one side to the other of the stope. When the stope breaks through into the heading above, the stubbing bar is shifted to the roof, but with no material effect upon the operation of the scraper.

This method may be designated as a single-necked stope but the tendency, in practice, is to drive two or more stopes, each being operated through a separate opening or neck, the stopes running together and forming a stope the combined width of them all. This method may then be designated as a multiple-necked stope. Stub pillars are usually left at both the top and bottom of the stopes and, if found necessary, small pillars may be left within the stopes for protection against caving ground. As an extra precaution, wide pillars are being left between stopes and are staggered between headings.

As stated before, it is important that the size of stopes does not exceed the safe working limit with respect to support. As usually developed in this district, stopes are approximately 50 to 60 ft. wide by 130 to 150 ft. long, and 200 ft. wide by 100 ft. long. This is assuming that the heading interval is 150 to 200 ft.; with a greater or smaller interval the length of stopes would be correspondingly greater or less.

As the preliminary stopes are narrow they may be considered as headings only, the stopes proper being run at right angles with them, consequently their width is parallel with the strike and their length parallel with the dip of the orebed.

Mining the Lower Bench

The lower bench of the Big Seam has been, and is still being, worked extensively in a few places. The localities where this has been done

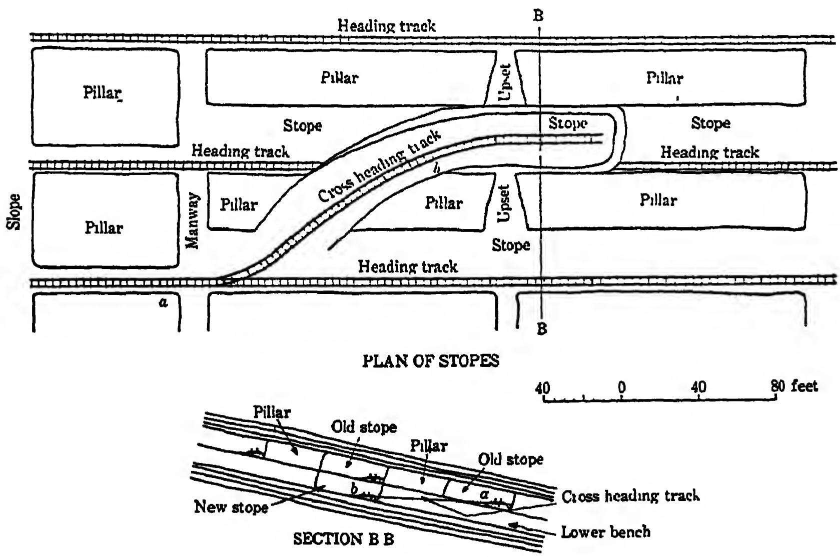

FIG. 10.—WORKING LOWER BENCH OF BIG SEAM BY CROSS-HEADING.

most successfully are noted for good conditions of top rock, so that large stopes may be worked with little or no support from props. Stopes are formed 30 to 35 ft. wide, 20 ft. high, and hundreds of feet long, with practically no support other than small pillars at the upper rib of the stopes or remnants of arch pillars. Many of these stopes have stood for years with slight indications of collapse.

First attempts at mining the lower bench, where mineable, were inadequate and unsatisfactory; but out of these developed the successful method now used (see Fig. 10). The mining of the lower bench or the upper bench stope bottoms in any particular stope is begun in the stope

immediately below, a cross heading being driven to the stope above. A branch track is turned off the main heading track, its grade being maintained as previously determined and as required by the dip of the orebed; it extends diagonally across the stope through the cross heading in the arch pillar to the middle or bottom of the stope above, where it is run parallel with the main heading track. The stope face worked in this manner is practically the full height of the lower bench, except where the dip of the orebed is high and the track is in the middle of the stope, when a wedge-shaped mass of ore will remain unmined at the bottom of the stope, just below the former heading. No ore is left unmined when the cross-heading track is at the bottom of the new stope, the stope having a bottom similar in position but 8 to 10 ft. below that of the original stope above. These stopes may be carried the full length of the stopes worked in the upper bench.

Underhand stoping is employed in mining the lower bench ore. The benches are usually 8 to 10 ft. high. Generally six holes arranged in three rows of two each are used; they are 10 to 12 ft. long, advancing the face by 8 ft. per round. The amount of ore broken per round in a stope 30 ft. wide and 8 ft. high is about 160 tons. Six holes are drilled in 8 to 12 hr. while shoveling is done on two shifts, three or four muckers working in a stope.

Removal of pillars is described later. Extraction of ore under favorable conditions when both benches of the Big Seam are mined and the pillars robbed will approach 95 per cent.; but it is rare that all conditions are favorable for such good work, and seldom can 80 per cent. be mined from the upper bench.

Mining the Upper and Lower Benches Together

This is a recent development of mining practice and is worked in part in only one locality. The lower bench ore here is of sufficiently high grade to be acceptable to the furnaces, so the whole orebed is being mined, although it is only an experiment at present. Development work and mining are first done in the lower bench, then, beginning at the breakthroughs connecting the rooms with the upper headings or mined ground, the pillars will be worked by taking one-half on either side of the rooms, the operations in adjacent rooms removing all the ore in that pillar. No two faces will be worked in a continuous line, but the working faces of the adjacent rooms will be arranged in step fashion, the most distant being attacked first; each adjacent face is 15 to 25 ft. in the rear of the previous one. The work of removing pillars will be done, as in all previous work, in the lower bench.

Following the removal of pillars and the placing of temporary supports, such as props, the upper bench will be attacked and blasted until

caving starts. The broken ore will fall on to the stope floors and be loaded into cars operating on the original room tracks. The caving of the upper bench will follow the removal of the pillars in the lower bench, the ore being removed until the presence of waste rock indicates that it is too lean for use. The work as planned is a panel system (all available ore in a panel is removed, the top rock caves and fills the worked-out area) and a number of advantages will result therefrom, such as: control of caving ground, high percentage of extraction, and low costs.

Robbing Pillars

In regular stoping operations, with heading intervals of 65 ft., an average of 60 per cent. of the ore in the upper bench is removed. This may be termed "first mining" while the robbing of pillars may be termed "second mining." Pillar robbing is the final operation in the removal

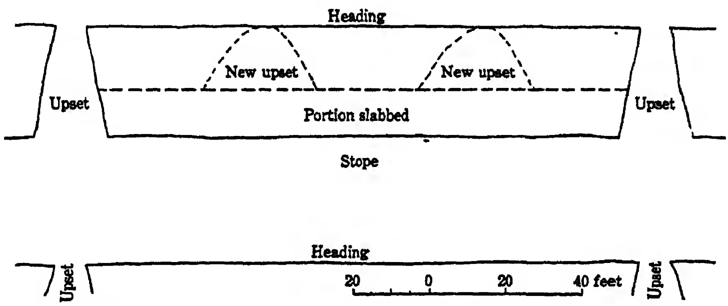

FIG. 11.—REDUCTION IN SIZE OF PILLARS BY SLABBING AND DRIVING UPSETS.

of ore from the stopes and on its success depends the ultimate extraction. It requires more skill and care than regular stoping work, and still further weakens the top, so robbing is begun at those points furthermost from the main haulageways or slopes.

Where roof ore is left as support to the draw slate and it can be removed with safety, usually the first operation is to quarry or shoot it down; following this the other robbing work, such as slabbing and splitting of pillars and forming break-throughs, is done. Slabbing is generally started at upsets or break-throughs and is extended to adjacent upsets (see Fig. 11). Upsets are driven through the pillars, after slabbing has reduced their width about one-half, thus breaking the long arch pillars into a series of small pillars, irregular in size and shape.

Splitting pillars is often done by driving headings through them parallel with their longer dimensions, and breaking through the flanking pillars, thereby subdividing them into a number of small pillars. As a rule, the pillars to be robbed by splitting are laid off into sections, each of which is served by a branch or cross-heading track from the main heading track. Fully 50 per cent. of the ore in a pillar can be mined by

splitting, especially if carried on in conjunction with driving upsets. Splitting of pillars is extensively employed in working irregularly shaped areas resulting from change in dip of the orebed, the main difficulty being to lay track to the pillars with grades suitable for handling the ore. Gravity planes have been used, but switchbacks are preferred when practicable.

The robbing of pillars, including the entire thickness of the Big Seam, as when the lower bench is worked, is relatively much easier than robbing in the upper bench alone; there is a large surface to attack and the pillars have stood for some time under a greatly increased roof pressure, which has weakened them to a considerable extent. The two most common methods employed in robbing pillars following removal of the lower bench ore are by upsets and cross headings.

Breaking down pillars 10 to 15 ft. across and 18 to 20 ft. high is readily accomplished by placing two shots in the base and one two-thirds up the pillars. The amount of powder used need not be large.

Drilling and Blasting

Probably the presence of more or less prominent bedding and joint planes exerts the greatest influence in breaking ore in development and mining the Big Seam. The bond between the upper and lower benches, or between the benches and the parting material, is exceedingly weak and no difficulty is experienced in separating the ore from the slate. Aside from seams and slips, the ore is remarkably uniform in texture throughout the full thickness of the bed, and breaks with the use of a moderate amount of explosive when shots are placed to take advantage of the shape and condition of the face of ore worked.

A series of two holes each are generally employed in development and mining, these being placed in a practically vertical position. When the dip of the ore is low, the upper holes of the series are dry (and are usually so in any case), while the lower holes may be wet; but with moderately high or high dips they are dry. Holes vary in length from 4 to 12 ft. The upper and lower holes are directed toward the top and bottom of the orebed, but are seldom drilled closer than 10 to 12 in. to the enclosing rock formation. The holes vary from 2 to 2⅛ in. in diameter.

Drills are mounted on tripods, no columns being employed, the two holes of a series or unit being drilled, when possible, from one set-up. As a rule, holes are drilled so that shots may be placed back of slip planes, never in the slip itself. The more common practice is drilling holes dry. Water drills are employed only in narrow work, particularly in driving slopes and manways and in turning off headings. While water drills are not generally used, they are gaining in favor and one mine uses them entirely.

While a large number of piston drills, mounted on tripods and using solid steel, are used, the latest practice is the water Leyner type of hammer drill. The common makes of drills and air-compressors used are of the Ingersoll-Rand, Denver Rock Drill, Sullivan & Chicago Pneumatic Tool. A few hand hammer drills are used for small work. Air is supplied at about 80-lb. pressure.

Explosives used are generally 60 per cent. dynamite in narrow work and 40 per cent. in breaking ore. Changes vary from 8 or 10 to 16 sticks for upper holes, and from 12 or 14 to 16 or 18 sticks for lower holes; the latter are usually fired first, hence require the heavier charges. Fuse and caps are used in firing all shots, except in slopes and manways, where electric blasting is practiced. Cartridges are 2 in. in diameter.

Support of Mine Workings

The support of underground workings in the mines of Red Mountain has, with few exceptions, been a matter of small concern. With increased depth of cover as the workings are extended and with the growing tendency to leave smaller areas of unmined ore as support, there will come a time when much trouble will be experienced in providing adequate support for the workings. In fact, in those mines where operations are carried on at a vertical depth of 1500 to 2000 ft., considerable trouble is already being experienced, and with greater depth the problem will become more serious.

The local problem of supporting workings is complicated by the not unusual condition of occurrence of water-bearing formations overlying the orebeds. The distance of the principal water-bearing formations (Fort Payne chert and Hartselle sandstone) from the orebed is approximately 150 and 300 ft. respectively. The character of the formations lying between the orebed and the water-bearing formations prevents the downward movement of any considerable amount of water into openings made in the orebed; but the breaking of the intervening impervious beds by fissures caused by subsidence permits the drainage of the reservoir formations and the entrance of water into the workings. Already, millions of gallons of water are being pumped daily from the iron mines, which emphasizes the importance of preventing falls that would tap the overlying water-bearing formations.

A careful study of the ore in the bed worked and of the roof or top formations reveals the strength and weakness of the elements affecting support in the mines. The formations overlying the orebed are composed of shales (slates) and sandstones, which are strong and would as a rule give adequate support for any reasonable size of workings in the orebed, but their strength is materially reduced by lines of weakness. Both the orebed and the top formations are cut by cleavage or slip planes, which extend for some distance above and below the bed. The orebed is then

broken by the well defined slips, which reduce its strength when formed into pillars; in like manner the top formations are seriously weakened—the compressive strength of the ore being diminished in the former instance and the bending strength of the top formation in the latter. As a further source of weakness, the top rocks are cut by cross-bedding planes, which often reduce the length of what would otherwise be long and strong beams of rock to those with lengths only a few times greater than their thickness. While cross-bedding planes are of common occurrence, they are not as persistent nor as prominent as are the slip planes.

The dip of formations has a decided influence on methods of working as well as support. With flat-lying beds, the entire weight of the cover must be supported by pillars or timbers; while with dipping formations, the vertical component of the load to be supported falls more or less obliquely across the pillars until it passes entirely into the walls when verticality is attained.

The iron orebed is cut by numerous faults, a few of large displacement, but by far the larger number of only several feet of displacement. From the standpoint of support, faults have small influence except locally, when the top formations may be badly disintegrated and weakened. The occurrence of several lines of faulting and displacement at intervals of a few feet, and the presence of considerable quantities of water, usually produce bad conditions of support, but normal conditions may prevail in the adjacent headings both above and below. As a rule, fault planes are tight and little water finds its way into the workings through them; however, there are instances where considerable water enters directly through or adjacent to them.

Character of Support Employed

Although ore, when left in pillars, constitutes the best possible means of support, the urgent demand of more ore and, at the same time, a demand for higher extraction tend toward a further reduction in size of pillars. Observation in the mines has shown that for moderate depth of cover, adequate support can be secured if 30 per cent. of the ore is left unmined, which is the practice in a number of large mines.

Supplementing the pillars of ore as support, the use of roof ore is an important adjunct. The thickness of this ore varies widely, but where left as a temporary or permanent top rock, it averages probably 10 to 12 in., the range being from a few inches to 3 or 4 ft. The strength of the roof ore, as in the case of ore in the pillars, is materially weakened by slips and cross-bedding planes, but it may serve a useful purpose as a temporary support to the weak top rock.

Barrier pillars are left between properties; they vary in width from 50 to 100 ft. They are supposed to be unbroken strips of unmined ore, but are usually cut at frequent intervals by headings.

Props have been used as roof support since mining began in this district, but in many instances they were far too numerous and too closely spaced; it is claimed that the work of setting timbers—wedging them and pegging—is in itself largely responsible for failure of top rock. At present, few props are used, and they are not placed closer than 50 or 60 ft. of the stope face; nor are they pegged (see Fig. 12).

Special forms of support, such as sets of round and sawed timber, crib work above sets, steel rails, and pack walls, are used, particularly in slopes, headings, tunnels, etc., but as a rule their use is confined to localities in which bad conditions of roof exist. Concrete is now being used satisfactorily in slopes.

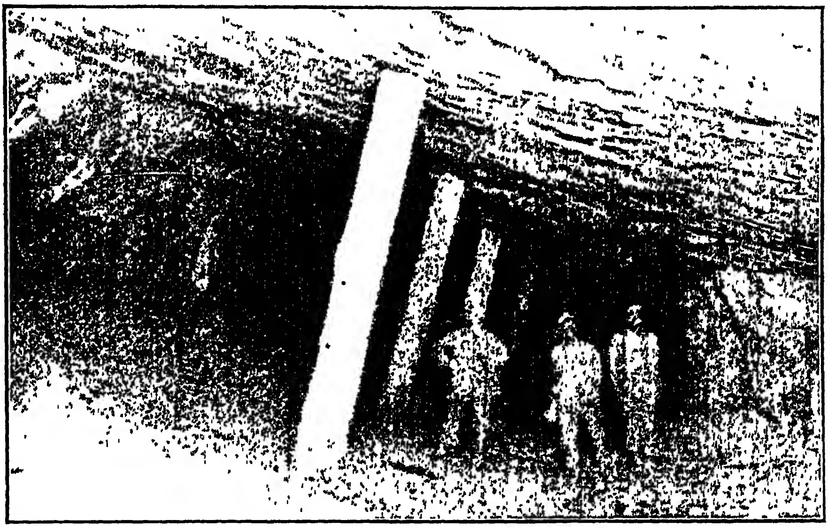

Fig. 12.—Support of roof in stope by props.

While props as a rule do not break under excessive weight of roof, sets do. The failure of pillars has not been a matter of much concern; but with increasing depth of cover it will demand serious attention and may require radical changes in mining practice. That pillars are failing under pressure of cover has been proved by extended observation. Probably most of the pillars that were observed to be failing show a more pronounced disintegration at the bottom or foot, which in many cases is so intense that the action assumes the form of underhand stoping. The failure of pillars takes place irrespective of direction, occurring on all sides of pillars alike, but it is more pronounced on the lower side and in upsets, being rarely observed in intensity on the upper side. The shape of the pillars seems to be the controlling factor in direction taken by the lines of failure, although slip planes have a marked influence.

The phenomenon of explosive rock, which has been observed in certain copper mines of Michigan and in the gold mines of India and the Trans-

vaal, has been observed in the limestone and sandstone above the orebed of the Big Seam. While the cause is not definitely known, it is probable that the distortion of the ore under pressure is the cause of this explosive action, and the spalling of pillars may result from the same action but differs in degree only from explosibility.

Handling Ore in the Iron Mines

The dip (true average 20°) of the orebed presents ideal conditions for handling ore in the stopes, the higher and lower dips requiring slight modifications in the methods commonly used. The actual handling of ore in the stopes is done by hand and by mechanical loaders or scrapers; the former is the more common, although large tonnages are handled mechanically. The cost of mechanical loaders is probably somewhat higher than that of hand shoveling, but undoubtedly it will be reduced to that of the latter and eventually will probably be much cheaper.

Ore, when blasted down at the face of stopes, is brought to the level of the heading track, where it is reduced in size by sledge or by blockholeing (blasting), and is then loaded into cars for transfer to the slopes.

Wooden cars holding 2 tons are used, but they are generally "trimmed" or "racked" to hold 2½ tons, which is not good practice. An all-steel car of 2-ton capacity is now being used extensively to overcome the disadvantages attendant on loading by hand; to take care of the mechanical loaders, 5-ton steel cars have recently been installed.

Under normal conditions, a man will shovel 10 to 13 tons of ore per 8-hr. shift, three shovelers in a stope handling 35 to 40 tons. These men also tram the loaded cars to slope stations, the empties being returned by mule. The shovels are square-nosed, weigh 5½ lb., are 3½ ft. long, and the load lifted averages 20 lb. The loaded cars are transferred by gravity to the loading stations at the slopes in twos and threes, the shovelers riding them.

Motor haulage is not applicable to heading work because of the restricted range of operations; yet if a motor could be made to serve a number of headings by transference on a slope, it might be a practical method of procedure. At present, motor haulage is confined to transferring ore from more or less distant points, such as between areas worked by separate slopes, which are connected by crosscuts made in the rock and driven on a grade slightly in favor of the loaded cars. The application of motor haulage to practically horizontal areas is occasionally employed.

When the distances through which ore had to be handled became too great and the stopes too narrow for branch tracks to be employed to advantage, shaking conveyors were installed and, for a time, were extensively used; they have been abandoned.

When the heading interval reaches 80 ft., branch tracks are applicable; but it is more usual to employ gravity planes, thus permitting the tracks to be laid directly up the stopes and taking full advantage of the dip (see Fig. 13). Cars operate on both tracks, empties being delivered to the working faces, and loaded cars returned to the heading tracks. At some intermediate point between the two pulleys, but generally near one of them, is a brake by which movement of the cars may be controlled. The balanced or gravity planes have had extended application in certain mines because of the elasticity of the system, but they are being replaced by mechanical loaders or scrapers. The use of balanced planes in large stopes is shown in Fig. 8.

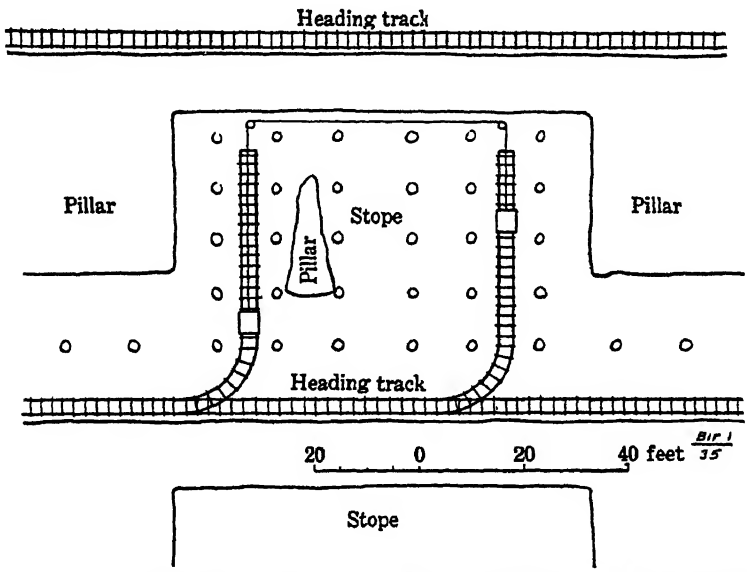

FIG. 13.—USE OF BALANCED OR GRAVITY PLANES ON MODERATE DIPS.

The mechanical loaders or scrapers move the ore from the stopes to and over apron plates placed at the necks of the stopes and discharge it into mine cars, which in turn are run to the skip-loading stations at the slopes. The scraper drag line is operated by a motor car having electrically driven drums, which also move the cars. While the use of mechanical loaders is comparatively recent, the success attained has justified their application and doubtless their use will increase in the future. Their application to all conditions of dip and top formations has not been made, and there may be limitations, involving changes in method of mining and support. These loaders reduce the cost of development work and increase the output from two to three times that of hand work.

Cross-headings are used extensively in working the lower bench, the cross-heading tracks being turned off the main-heading tracks at such points along the headings as seem desirable, and are run at such grades as will permit cars to be operated, if the dip permits, along the middle of the stopes above, but on the foot wall of the Big Seam. On high or moderately high dips, the cross-heading tracks may not extend to the middle of the stopes above, but are placed on the foot wall immediately below the former heading tracks above. It is obvious that on high dips cross-headings have limited application, but with an average dip of orebed they can be used to advantage (Fig. 14).

Probably one of the most extensive applications of cross-headings to the handling of ore is in the robbing of pillars, in which use they are

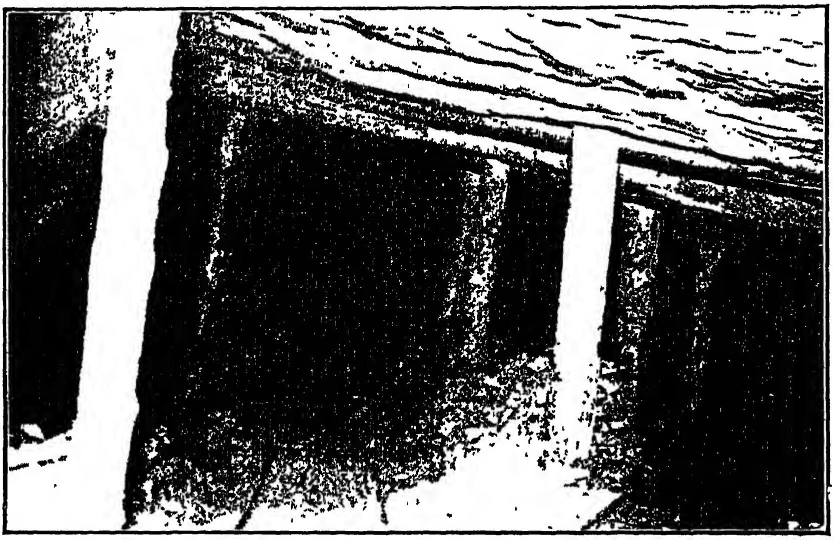

FIG. 14.—CROSS-HEADING EXTENDING THROUGH PILLAR.

seldom extended beyond the heading next above, but reach and are continued along the center line of the pillars. It is but one step from the use of cross-headings in robbing pillars to their application to mining the lower bench of the Big Seam, and at the same time robbing pillars in both the upper and the lower benches.

Cross-heading tracks are often extensively used in reducing the size of pillars by slabbing from the lower side, which work may be extended, with small heading intervals, to the removal of practically the whole of the pillar.

Mules are employed in hauling ore cars on the heading tracks and returning the empty cars to the stope faces. In the smaller mines, the mules are usually taken in and out each day; but in the larger mines, they are kept underground for several weeks or more.

In slopes, with car or trip haulage, no orebins or loading pockets are used; with skip haulage, there is either an arrangement for dumping cars by hand or by air-operated tipples into the skips below, or an orebin or pocket of several hundred tons capacity is placed at the foot of the slope, above which pocket is a rotary dump of 4 to 6-car capacity. The speed of loaded trips of cars to the surface ranges from 800 to 2000 ft. but averages 1400 ft. per min. Skips hold from 10 to 12 tons, but seldom carry more than 7 or 8 tons. The rope speed for skips is from 1500 to 3000 ft. averaging probably 2300 ft. per min. On account of variations in the grade of slopes the ropes often strike the top and bottom and are thereby damaged, but the installation of rollers has reduced the wear and tear considerably.

On the surface, trip haulage requires one type of rock house while skip haulage requires one of wholly different form; that for the former is a long comparatively low structure to permit trips of cars being run upon them from the slopes and disconnected from the haulage rope, while skips operate on tracks supported above the ore pockets and are part of the rock house proper. The grade of skip tracks on the rock house is practically the same as that of the slopes, although in most cases it is maintained at an angle of approximately 16°. At both trip and skip rock houses, the ore passes through gyratory rock crushers, then to railroad cars, and finally to the blast furnaces.

Drainage of Iron Mines

Aside from the influx of water through breaks and cave-ins extending to the surface or to certain water-bearing formations overlying the ore-bed, the amount of water entering the workings is relatively of slight importance, but the condition is growing worse and will become a menace to mining operations. Surface water can be largely eliminated with due care and attention, therefore the main problem is the reduction in the amount of water coming from caved ground. It is evident that the solution of the drainage problem depends largely on the method of mining employed, which in turn is dependent on support of workings.

Water entering the mines is caught direct in sumps or by catch basins and transferred to the sumps, which are stopes closed with concrete stoppings or dams. Where a number of slopes are operated together, it is customary to collect the water from all at one point where the pumping plant and main sump are located. The main sumps are usually placed just below the wet area, which gives the minimum lift for the bulk of the water entering the mines. The water entering the mines from the main sump to the foot of the slope is handled by one or more pumps that discharge into the main sump, the main pumps completing the lift to the surface.

The pumping equipment consists of gathering pumps for handling water from isolated basins or pools to sumps—as a rule their capacity is not large; small sump pumps to lift water to main sumps, as from main slope and auxiliary sumps; main sump pumps which eject water from the mines.

The gathering pumps are usually of the direct-acting or centrifugal types operating under heads of 200 ft. or thereabouts—Cameron, Aldrich, and Worthington pumps are commonly used. The capacity of these pumps varies from 50 to 100 gal. per min. They are driven by compressed air and electricity.

The second type of pumps may be designated as station pumps and belong to the centrifugal and multiplex types, although some direct-acting plunger pumps are used. Double-suction, single-stage pumps directly connected with motors are usually employed. Triplex and quintuplex, single-acting plunger pumps, motor driven through gears, are also employed as auxiliary pumping units. These pumps generally operate under heads of 100 to 500 ft. and have a capacity of 100 to 500 gal. per min. Aldrich, Worthington, Allis-Chalmers, Prescott, and Cameron pumps are common types used.

The main station pumps are practically universally motor driven, practice favoring the centrifugal type. These pumps are single- and multi-stage, direct-connected; four and six-stage types are common. The capacity of these pumps ranges from 800 to 1800 gal. per min. and they operate under heads of 800 to 1000 ft. The principal makes of main station pumps are Worthington, Allis-Chalmers, Aldrich, Prescott, and Cameron. Discharge pipes range in size from 6 to 10 in. in diameter and are laid in manways.

Ventilation of Iron Mines

As mining on Red Mountain has gradually developed from surface to underground workings of considerable extent, the need of ventilation has not been felt as it would had the mines been developed by vertical shafts from the beginning of operations. Further, the use of compressed air in drilling has delayed the application of positive methods of furnishing fresh air to the workings. But the natural movement of air, even though assisted by air discharged from drills and hoists, is inadequate at best, and in a relatively short time ventilating equipment will have to be provided in the more extended workings. Small blowers or fans are used occasionally, forcing air into dead ends through canvas or metal piping, but their use is limited and has been studied[3] by the Bureau of Mines at Butte, Mont.

[3] G. E. McElroy, and A. S. Richardson: Experiments on Fan-pipe Installations at Butte, Mont. Reports of Investigations, Serial No. 2509, Bur. of Mines (1923) 44.

Factors affecting the atmospheres of the mines are ore and rock dust from drilling and blasting, decaying timber, and fumes from explosives. Air analyses from working places reveal relatively high percentages of carbon dioxide, carbon monoxide, and dust, although usually the oxygen content is not abnormally depleted. The main remedy for these unfavorable atmospheric conditions is the circulation of fresh air to remove gases and dusts and to provide comfort against high temperatures and humidity; the only positive method of insuring this condition is the circulation of air by mechanically driven fans on the surface, supplemented by air control, such as doors, brattices, regulators, tubing, and so forth. Also, the adoption of vertical shafts in future development, with the application of fans and definite methods of distribution of air to the workings, will eliminate present atmospheric conditions and result in great improvement in working conditions.

Suggested Changes in Mining Practice

In the methods of mining discussed in the foregoing pages, including development, working, support, drainage, and ventilation, previous and recent practice are given, together with methods employed in the transition between the two. It now remains to outline briefly possible future practice, based on past experience in the mines and on conditions that it is reasonable to assume will obtain with the extension of the workings to greater depths. However, the conditions under which work will be carried on when the operations have reached greater depths are not based wholly on assumption, but on known conditions. The conditions referred to may be considered as definite and suggestive; the former are noted in the present workings developed by slopes, the latter are observable in present deep workings, though limited in extent.

The conditions observed in the operating slope mines cover a wide range both in lateral extent and in depth of cover, and show the changes that are taking place progressively as the orebed takes on more weight from superimposed formations. It is but a step, therefore, from observed conditions in the mines approaching considerable depth to conditions in mines that have already attained considerable depth of cover; the facts in the former case are manifold while in the latter case they are limited. In view of the known facts and those indicative of probable future conditions, tentative suggestions relative to later practice are here offered; however, changes in practice should be made, and will probably be necessary in the not distant future, particularly in certain localities. What is needed is not more mines but better operated and equipped mines and more efficient, low-cost producers; it is with these considerations in mind that the following suggestions are made:

The problems confronting future mining operations are threefold; namely, subsidence of the surface, caving and fissuring of top rock, and

conditions affecting full extraction of the ore. It can hardly be said that any one problem is of more importance than the others, because in one locality one problem may assume serious proportions, while in another locality another problem may transcend the others in importance and thereby demand special consideration.

Subsidence of Surface

It is doubtful whether the subsidence of the surface will affect other than residential property, and will be confined to a comparatively narrow strip of land lying on the eastern and southern slope of Red Mountain, particularly that part immediately adjacent to the city of Birmingham. Probably less than 50 per cent. of the orebed has been worked underground in this locality, consequently the surface can be given such protection as is possible by future mining. In those localities where mining has already been done, the subsidence of the surface cannot be controlled except at considerable expense.

Outside of the residential districts, the land is not of much value except for grazing purposes, therefore its subsidence will be of no serious consequence; and being owned or controlled by the operating companies will probably not result in legal complications. Further, it is probable that subsidence will not extend much beyond the slope of Red Mountain, for heavy beds of limestone and sandstone occur which will check any pronounced and serious irregular settlement of the formations overlying the orebed. If this is the case, there will be no serious subsidence in Shades Valley beyond Red Mountain.

Control of Influx of Water to Mine

While failure of top formations resulting from working the orebed will take place and reach to considerable vertical distance, the most extensive fissuring and caving will probably take place within the zone of leaching of certain formations above the orebed, and will correspond to but extend farther from the outcrop than the leached zone observed in connection with the orebed. However, extensive caving has taken place in a number of localities, the breaks in the top formations reaching to and disturbing certain water-bearing formations, particularly the Fort Payne chert.

The distance from the outcrop of the orebed on Red Mountain to where the chert bed ceases to be sufficiently porous to hold water in any considerable amount has an important bearing on the problems of mining and drainage. Information regarding the extent of the leached zone of the Fort Payne chert bed is not available; but should it prove to be extensive, the question of adequate support in that area will have to be given serious consideration. Preliminary mining operations have probably extended over the larger portion of the leached area of the Fort

Payne chert formation; and except in those localities where robbing has taken place, there is reasonable security against tapping the water-bearing beds through caving and fissuring. Beyond the leached area of the Fort Payne chert bed, the water problem is not a menace as a result of caving of top rock and as a source of additional water entering the mines. Continued and extensive robbing of pillars in the old workings means more and larger breaks, tapping the water-bearing formation and admitting an increasing volume of water to be pumped from the mines. Protection must be provided or a limit will be reached beyond which the mines cannot be economically freed of water; the protection furnished must be one of two forms or both as conditions dictate—namely, for virgin ground, preliminary or one-stage mining alone, and for previously worked ground, no robbing of pillars, or if so, extreme care should be taken in the work. In other words, all mining work must be limited to a definite percentage basis of unmined ore. If this practice is followed to the lower limit of the leached zone of the Fort Payne chert bed, ample protection will probably be provided against any material influx of water through caving ground.

Extensive breaks in top rock can be limited, and the damage already done can probably be remedied to a large extent by grouting the known breaks or by isolating the caved areas by walls of concrete; both methods, while relatively expensive, would greatly reduce the present operating cost, to say nothing of guarding against materially increased difficulties. Grouting might, in some cases, prove to be the more satisfactory as in some localities the top rock permits the passage of water through it and around dams and barriers built in the stopes and headings.

As a precautionary measure, guarding against breaking and fissuring of the top formations, extensive caving in stopes can be prevented by systematically attacking the roof where signs of failure show and causing it to fall in such a way as to fill the stopes with the broken rock. By this method of procedure, it should be possible to limit the height of cave-ins to considerably less than 100 ft., which would prevent the fracturing of the water-bearing formations.

Changes in Development

While there is small prospect for any material change in the near future in the method of developing the iron-ore mines, it is not unlikely that a change from slopes to vertical shafts will become increasingly desirable and ultimately will be required. Standardization in equipment and practice, already applied to many of the mines and, to a more limited extent, between mines of different companies, while desirable, acts as deterrent to the adoption of new methods. This is particularly true with development, where a change from skip to cage haulage would mean the practical scrapping of or a greatly restricted use of the present

slope equipment. However, the facility of handling ore underground greatly increases production, and the economy of operation of vertical shafts over slope haulage would be so pronounced as to remove all doubt as to the desirability of a change being made.

Changes in Mining

The problem of future working of the mines resolves itself into the adoption of methods of mining that will permit the maximum extraction of the ore with minimum loss through caving. The solution of the most serious problems lies in the future; the conditions between the present and future mining require immediate consideration, and their successful solution will insure equal success of the more serious problems to be met later. The work of mining the ores of Red Mountain is in a transition period between definitely known and only partly known conditions, with a water menace behind.

With a wide range of conditions of support occurring in the district, it is not possible to outline methods that might be applicable in each case, but a general statement will suffice. In the first place, future workings will be under greatly increased cover and pressure of cover will increase progressively, which increase will make itself felt in a much more positive way than similar increases in the past, the accumulated load being vastly greater. Consequently, the size of stopes formerly considered safe may be beyond the safe working limit, and areas of pillars known to have a fair factor of safety may prove to be inadequate for future work.

The present practice, fairly extensively followed in the larger mines, is to leave 30 per cent. unmined ore in the pillars while apparently ample; the pillars may have to be increased even to 50 per cent. or more, in future preliminary work. The final removal of ore by drawing the pillars may increase the extraction to 85 per cent. or more, but must be removed in such a manner as not to jeopardize the adjacent workings by caving and crushing of the pillars.

The practice that permits caving of top rock in large stopes may prove satisfactory under comparatively light covers; but in the deeper workings will hardly be permissible even with wide barrier pillars, as crushing will undoubtedly take place and follow down the dip breaking the barrier pillars and spreading in all directions. The destructive action of squeezes crushing wide pillars left as protection and the ultimate wrecking of extensive workings is too well known to be discussed in this connection. In the anthracite fields of Pennsylvania, barrier pillars 150 to 200 ft. wide have proved ineffective in stopping, or even checking for any considerable time, the squeezes which developed and swept down the dip of the beds. It is possible, however, that the control

of isolated squeezes together with influxes of water through slip and fault planes might be successfully accomplished by enclosing pillars in connection with concrete stoppings.

Applicability of Caving Method

With the approach to areas of high pressure, some system of working the mines will have to be adopted. The controlling factors in making the choice should be experience in other districts where similar conditions prevail; at the same time the method adopted must be sufficiently elastic to meet varying conditions and be economical. As the pillars and top rock will be difficult to maintain, it would be logical to choose a method that would permit of their collapse at an early period in the operation in order that costs of temporary support even would be relatively slight. Briefly outlined, a caving method might be adopted to advantage, the preliminary work such as headings and stopes being carried narrow in order to reduce the need of support and, where possible under roof ore, the final operation removing the pillars and roof ore and permitting the top rock to cave. By carrying the stoping faces in a direction that cuts across the slip planes at an angle of 45°, or thereabouts, the greatest strength of pillar and top rock would be developed; and by forming the face into a straight line in order to prevent concentration of pressure at any one point, the control of the caving top rock could best be secured.

With a change in the character of ore in the lower bench of the Big Seam, or through the successful solution of the beneficiation of the high-silica ores of the lower bench, a new condition will have to be met in that the whole thickness of the Big Seam will be workable. In this case, the caving method suggested would be modified to meet the new conditions, the preliminary work being done in the lower bench, also the removal of pillars, the upper bench serving as roof to such workings and caving as the pillars were drawn or robbed. The superior strength and thickness of the upper bench, compared with the weaker slate roof, would tend to insure the successful working out of some such method.

The mining of the upper bench or the combined upper and lower benches of the Big Seam by a caving method would, of necessity, be done by retreating from the limits of the property, and across the line of dip, to insure against squeezes starting and getting beyond control. However, the work could probably be done best by developing in panels, the enclosing pillars serving as a protection in the earlier stages of the work of mining. The details of such a method, if at all applicable, would have to be worked out after the conditions existing in the ore and top rock were understood. There is no doubt, however, that a method will have to be devised that can be widely used and controlled, which permits a relatively high percentage of extraction and that is economical in labor and supplies.

Change in Ventilation

The satisfactory ventilation of the average iron-ore mine has about reached its limit, while in a number of cases the limit has been exceeded and unsatisfactory conditions exist. It cannot be said that there is a complete system of positive ventilation in any mine in the district, the installation of blowers to supply air to the advanced workings and dead ends are inadequate, in that the source of air for such purposes has, in most cases, been rendered unfit by its passage through the workings prior to its delivery to the advanced workings. Carefully planned systems of ventilation are needed in all of the large mines if satisfactory conditions for working are to be provided. However, the installation of positive exhaust or pressure systems could be accomplished with no great expense in the present slope workings, although their installation in vertical shafts would be preferable.

Acknowledgments

Acknowledgments are hereby made to the operators of the iron-ore mines for their coöperation in the investigations that have made the preparation of this paper possible. Particular acknowledgment is due to C. E. Abbott, General Manager of the Tennessee Coal, Iron, & Railroad Co.; C. E. Bowron, Chief Engineer of the Gulf States Steel Co.; W. M. Lacey, former General Superintendent of the Woodward Iron Co.; W. J. Penhallegon, General Superintendent of the Republic Iron & Steel Co.; J. E. Strong, Vice-President and Chief Engineer of the Alabama Co.; H. J. Thomas, General Superintendent of the Sloss-Sheffield Iron & Steel Co.; as well as many others whose advice and suggestions have been invaluable.

Roof Support in the Red Ore Mines of the Birmingham District*

By W. R. Crane,† Birmingham, Ala.

(Birmingham Meeting, October, 1924)

The support of roof in mines is dependent largely on the character of the top rock and its occurrence. The formations overlying the orebed in the Birmingham district are sandstone and slate. The sandstone occurs in beds of sufficient thickness to constitute important elements in the system of support; the slate is relatively strong but is thinly stratified. The alternating beds of sandstone and slate furnish an excellent combination in that the former are strong and the latter are impervious; however, their continuity is broken by jointing or slip planes.

The thickness, and consequent weight of the overlying formations, is important in so far as the size and arrangement of pillars are concerned but may be relatively unimportant, temporarily at least, as a factor in the support of the roof. The position of the respective beds of sandstone and slate, their relative thickness, and their inherent weakness because of the occurrence of faults and folds, may exert a predominating influence on their support in place, which is affected only in part by the weight of the cover.

The only mine in this district that operates under a cover of approximately 2000 ft. (1900 ft.) is the Shannon so-called "twin slope," situated 14,000 ft. southeast of the portal of the No. 7 mine of the Tennessee Coal, Iron, & Railroad Co. The conditions in this mine indicate what may be expected in other mines when the same depth is attained; in fact, the effect of pressure is shown in several mines that have not reached two-thirds of that depth. It is evident, then, that conditions affecting the support of roof will not improve but rather will become more difficult with the extension of the mines into the valley under a constantly increasing weight of cover. Further, the difficulties will probably be augmented by the occurrence of faults and other disturbed ground, the presence and position of which at present are largely unknown.

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† Superintendent, Southern Mining Experiment Station.

Conditions Affecting Support

The conditions of occurrence of ore and top rock affecting support, given in their relative order of importance, are as follows:

1. Varying thickness and amount of sandstone and slate in the top rock.
2. Presence of slips and bedding planes, regular and cross bedding.
3. Chemical and physical character of ore, inclusions of masses of lime and slate, pebbles, etc.
4. Presence of rolls and pots.
5. Presence of faults and the folded condition of orebed.
6. Dip of orebed.

It is well established that strong thick beds are controlling factors in the support of roof formations in mines, while the weaker and thinner beds must be given support, both temporary and permanent. Under normal conditions, the width of workings is determined by the thicker and stronger beds, if close to the bed worked, the support of the thinner and weaker beds alone receiving attention. The amount of slate in the top rock and its distribution throughout the vertical height of the overlying formations is of prime importance in the support of the roof of the iron ore mines, the strength of the roof varying largely with the distribution of slate, as when interstratified with the thicker and stronger beds of sandstone.

As a result of earth movements that have disturbed the orebed, particularly with respect to dip and strike, the ore and the overlying formations have been broken by a series of remarkably uniform and regular joints or slip planes. So prominent and persistent are the slip planes or slips that, in the early days of the development, the slopes were driven with them, but as the slips do not usually conform closely with the dip such method of procedure has been abandoned; see Fig. 1.

There are commonly two distinct or major lines of slips, which are at approximately right angles with each other; in certain localities, though, there may be several series of slip planes, each series crossing the others at different but more or less constant angles as shown by the following record for one mine: S 50°W, S 5°E, S 40°W, S 55°W, S 60°W, S 80°E, S 45°W, S–W, S 80°W, S 73°E, S 60°E, S 75°E, S 22°W. Although only one set of slips may be revealed by mining operations, the existence of others is evident on breaking the ore in blasting and by sledge work.

The major lines of slip planes are taken advantage of in the development and working of the mines, particularly the latter; by their regularity and persistency in direction, both laterally and vertically, they facilitate the breaking of ore and materially reduce the cost of mining. The advantage resulting from numerous and prominent slip planes subdividing the orebed into slabs and blocks that can be readily broken by

blasting, however, is more than offset by the weakening of the top rock by breaking the continuity of the beds.

From the orebed, the slip planes pass into the overlying formations or top rock and are known to extend to the Fort Payne chert, which lies about 150 ft. above the Big Seam. All formations associated with the orebed and including it, therefore, are extensively broken in two directions by the jointing or slip planes, which are of such prominence as to have

FIG. 1.—SLOPE DRIVEN ON SLIP PLANES.

a definite and serious effect, thus reducing their strength to a fraction of what it would otherwise have been; see Fig. 2.

Other factors, similar to slips in nature, that further weaken the ore and top rock are bedding planes, both regular and cross-bedding. Bedding planes occur in both ore and top rock. Normal or regular bedding planes, and to a less extent cross-bedding, in the ore present lines of weakness along which the disintegration of pillars may act, but it is to cross-bedding in the top rock that the chief element of danger must be charged; see Fig. 3.

The constituents of the ores on Red Mountain show wide but rather constant variations, iron and alumina being the most constant. The relation of the various constituents to porosity and specific gravity as well

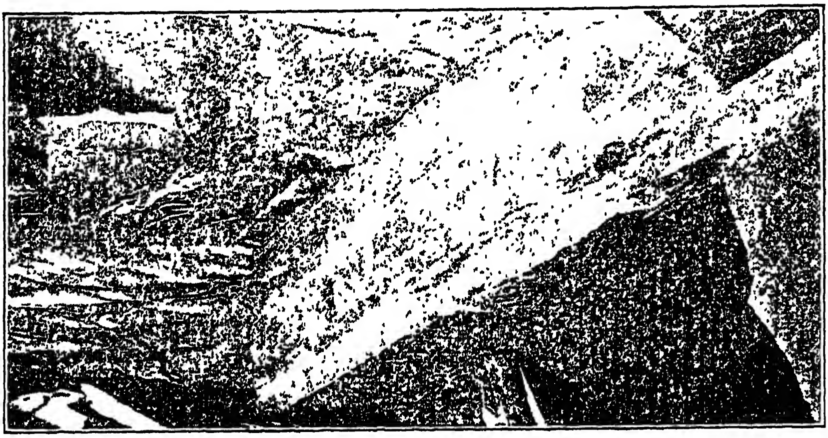


FIG. 2.—EXTENSION OF SLIP PLANES INTO TOP ROCK.

as their variations are shown in Fig. 4. The specific gravity of the ore varies directly and closely with the iron content and, to a less degree, with the lime content; porosity varies with silica and alumina or the insolubles. The range in constituents of the ore, according to the series of

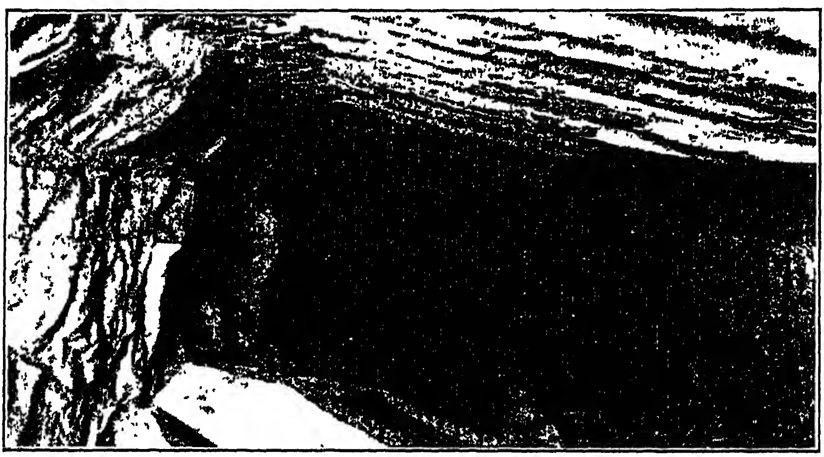


FIG. 3.—CROSS-BEDDING PLANES IN TOP ROCK.

samples taken, is as follows: Metallic iron 34.36 to 36.41 per cent.; lime 15.74 to 19.10 per cent.; silica 9.68 to 12.70 per cent.; alumina 2.75 to 3.38 per cent.; phosphorus 0.267 to 0.325 per cent.; sulfur, none.

The average porosity of the ore is 3.38 per cent., while the apparent specific gravity is 3.48; see Fig. 4. Fig. 5 shows the effect of the various

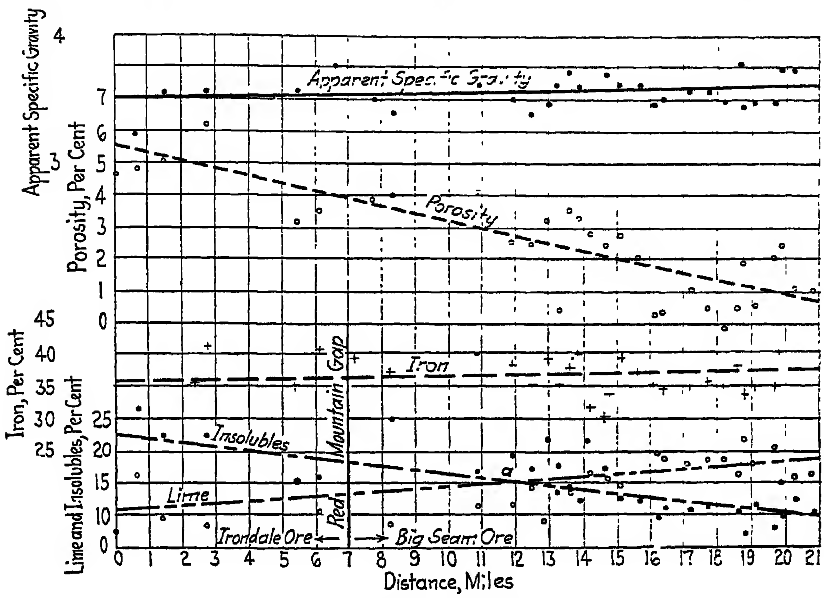


FIG. 4.—VARIATIONS IN SPECIFIC GRAVITY, POROSITY, AND CONSTITUENTS OF ORE.

elements on support, and covers a wide range of conditions from angle of break in top rock to thickness of roof ore and sandstone.

Other physical features also have an influence on the strength of the ore, when considered as an element of support for the roof. The effect of

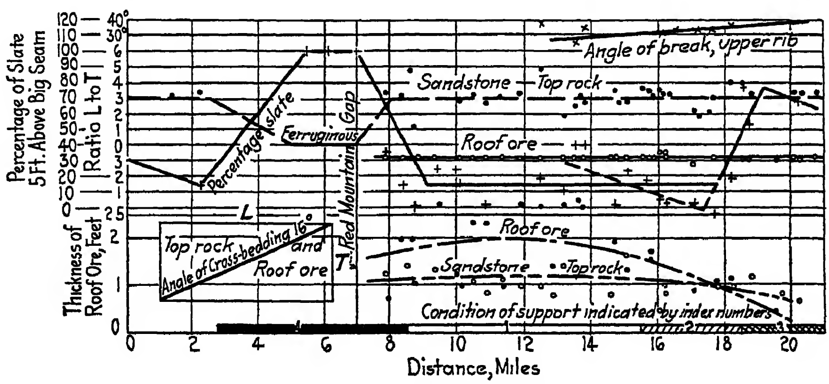


FIG. 5.—RATIO OF DISTANCE BETWEEN POINTS CUT BY CROSS-BEDDING PLANES AND TOP AND BOTTOM OF BEDS AND THICKNESS OF BEDS, THICKNESS OF ROOF ORE AND SANDSTONE TOP ROCK, PERCENTAGE OF SLATE IN FIRST 5 FT. ABOVE BIG SEAM, ETC.

pebbles and water-worn fragments of rock is much more pronounced than is generally believed, particularly when the ore is under high compression—failure of pillars is undoubtedly hastened by such inclusions. Masses

of limestone and slate in the ore of a pillar have an effect not unlike that of concentrations of lime which, by virtue of their difference in compressive

FIG. 6.—ROLL IN TOP ROCK IN ORE MINE.

strength, cause differential action between parts in the mass of ore and hasten disintegration.

Rolls and pots are of common occurrence in the iron ore mines, but are probably more common in the southern portions of the district.

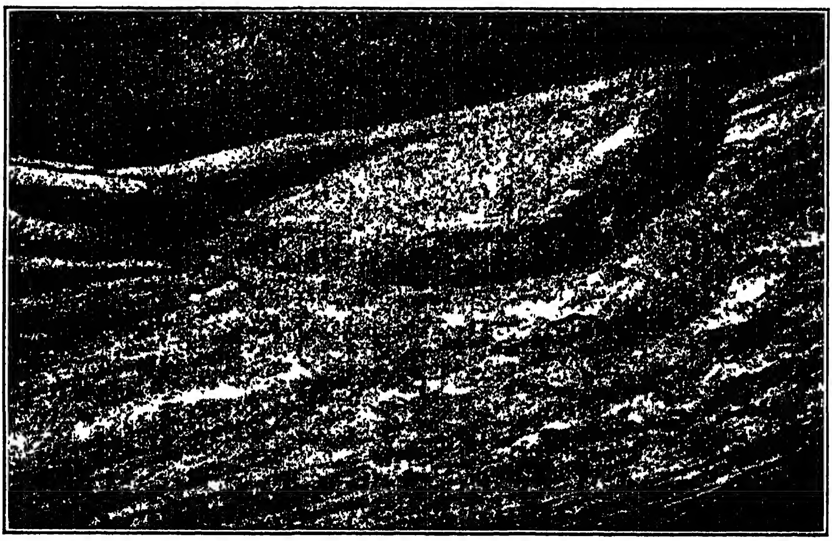


FIG. 7.—POT IN TOP FORMATIONS IN ORE MINE.

These irregular formations in the top rock may readily be confused with cross-bedding; in fact, the structure of both may be merged, the

resultant formation being even more treacherous than either of the merging forms in that irregular portions may become detached; see Figs. 6 and 7. The least that can be said of rolls and similar formations is that they indicate an irregular roof formation with high percentages of slate present. Under such conditions, narrow workings must be resorted to if temporary support of the roof by props is to be kept within economical limits.

Filled pot holes, in contradistinction to the pots and kettles of both coal and ore mines, are not uncommon varying from 8 to 25 ft. in diameter and are not usually cut out by the mining operations. They seldom seriously affect either the ore or the top rock, and falls of rock from the roof surrounding them are rare, because probably they

FIG. 8.—SLIP PLANES ADJACENT TO FAULTS.

generally decrease in size from below upward, thus causing the top to become tighter rather than looser with any downward movement.

The effect of faults on roof support in the iron ore mines of the Birmingham district is not fully appreciated and less well understood. The roof adjacent to faults is often seriously disturbed for some distance on either side of fault planes, which is shown by slips or secondary faulting. The slips may or may not show actual displacement; where they do not, they are similar to the ordinary slips caused by stresses set up by the down-drag accompanying faulting. The more prominent of these slips occur within 8 to 12 ft. of the main fault planes, the adjacent ground being disturbed and weakened. Further, the weakening of the faulting action undoubtedly extends much farther than is evident by slips, as is shown by the tendency for the roof to fall several hundred feet from the faults; see Fig. 8.

Investigation has shown that faulting has had a profound effect upon the physical condition of the top rock, particularly in certain localities,

causing dislocations both laterally and vertically. The formation of slips in the ore and top rock, with or without displacement, is particularly noticeable above the fault planes, although they may occur below the faults.

As the fault planes are usually approximately parallel with the strike of the orebed, the slips commonly have the same direction; it has, however, been observed that other slips occur nearly at right angles with the former and may be the ordinary slip planes made more prominent by the faulting action. In a number of these slip planes crossing the faults, lateral displacements have been noted in the direction of the dip; see Fig. 9.

In a number of localities, it has been observed that the down-drag of the fault has caused a slippage between the slate top rock and the orebed,

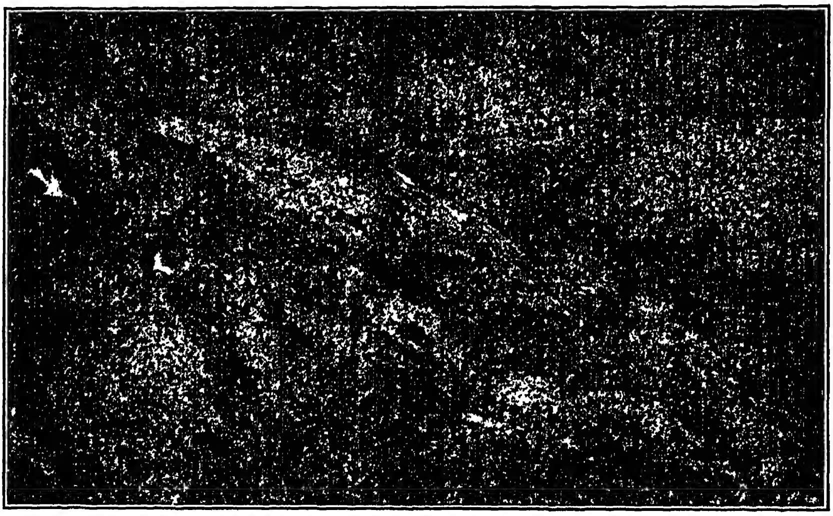


FIG. 9.—DISPLACEMENT IN SLIP PLANES PARALLEL TO DIP OF OREBED.

definite well-defined grooves resulting. This phenomenon has been noted several hundred feet from the fault planes and reveals the tremendous strain to which the top rock must have been subjected at the time of the faulting of the orebed and associated formations. That a differential action must have taken place between the separate layers of the top rock is hardly to be denied and a weakening effect on the bond connecting the layers must have resulted. That this is the case is evident from the weak condition of the roof in localities where evidence of the drag exist, which often falls in a comparatively short time after the removal of the ore, even in narrow workings; see Fig. 10.

Aside from increasing the prominence and extent of the slips, folding has not materially affected the problem of supporting the roof in the ore mines, except in those localities where close folding has broken the

top rock, which falls extensively when relieved from the supporting ore; see Fig. 11.

FIG. 10.—DRAG LINES IN TOP ROCK, LOOKING ACROSS THE DIP.

Under the normal dip of the orebed, no serious inconvenience is experienced in supporting the roof, but with dips of 35° and above, the tendency of the pillars to overturn presents an added difficulty to support.

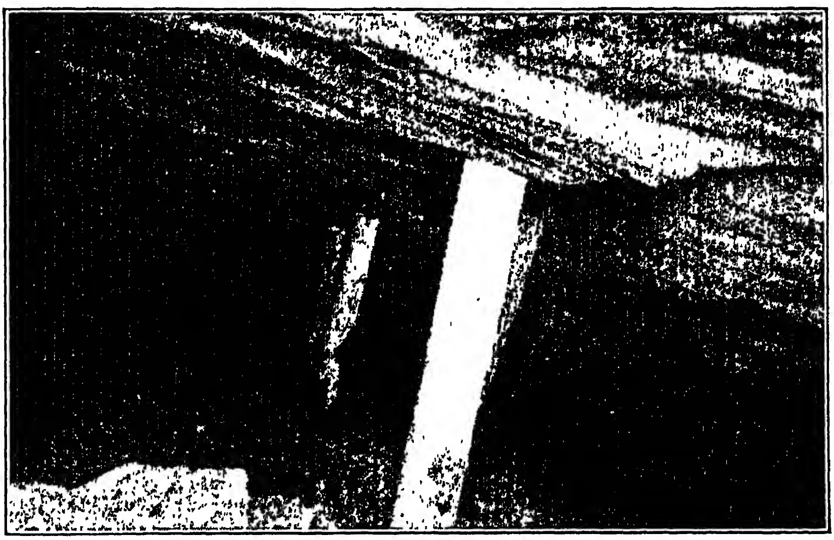


FIG. 11.—BREAKING OF TOP ROCK BY CLOSE FOLDING.

The tendency of pillars to overturn is shown by the development of shear lines within them and their ultimate failure thereby; see Fig. 12. Probably the steepest parts of the orebed have been reached and passed by

the mining operations and there will probably be a gradual flattening out beyond the flank of Red Mountain. In this case the normal condition of orebed with respect to dip will obtain and no radical change in method of support of roof will be required, except as the methods of mining change, increasing the width and possibly the height of working places.

FIG. 12.—SHEAR LINES IN PILLAR.

Of the factors, therefore, affecting roof support in the iron ore mines of the Birmingham district the most important are: (1) character and amount of sandstone and slate, (2) presence of slips and bedding planes, (3) irregularities in top rock as rolls and pots, and (4) effect of faults. Of these factors, the first two are of the most importance.

METHODS OF SUPPORT OF ROOF

Where the top rock is strong, the roof needs little or no support; but when weak, extensive timbering is necessary to maintain the top until the ore can be removed. With weak slate top, it is customary to leave unmined several feet of ore next to the roof which, if low in grade, may

remain as a permanent roof. The roof ore varies from 6 in. to 3 and 4 ft. and, unless broken and weakened by shooting, may stand almost indefinitely; see Fig. 13.

With dip of orebed varying from 15° to 25° and with normal occurrence of ore, the support of the roof in the mines, when worked by open stopes, is comparatively simple and easily accomplished. In the early days of underground mining, numerous props of good size were used, but this was found to be unnecessary and expensive. Former practice called for spacing of props from 12 to 25 ft. apart and often so close to the face that they were broken and blown out when breaking ore. In present practice, props are seldom placed nearer than 50 ft. to the working face,

Fig. 13.—Roof ore left as support to slate top.

except where conditions of roof render closer spacing necessary. The "pegging" of props to prevent their displacement by blasting has been practically abandoned.

The usual practice in timbering is to use medium-sized props, with or without pack-wall support, depending on the availability of waste rock. The props are usually 8 to 14 in. in diameter and are spaced 25 to 50 ft. apart. This use of props is largely temporary in character, being employed mainly in the support of draw slate; the props in reality, therefore, do not take much weight. There are, however, many sections of the district where little or no support is required; hundreds of feet of stopes are standing without support for the roof and apparently are no more in need of support than when the ore was removed years ago. On the other hand, extremely bad conditions prevail in many sections, where narrow

stopes and extensive timbering are required to maintain the stopes until the ore can be mined to the desired distance from the slopes; see Fig. 14.

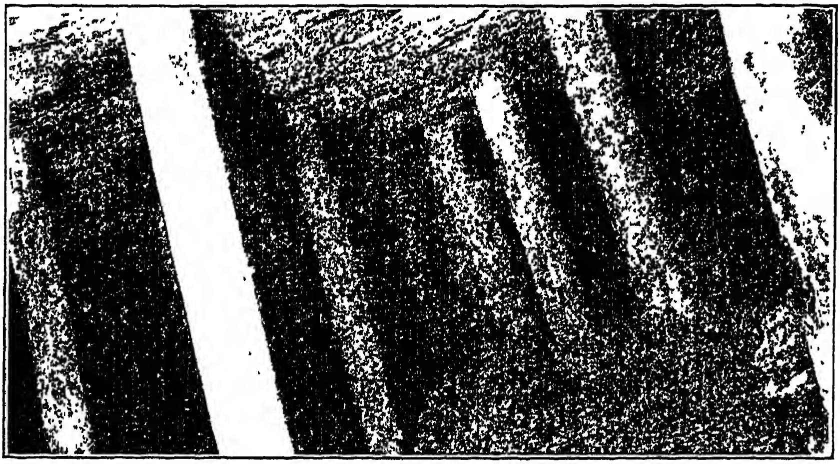


FIG. 14.—HEAVILY TIMBERED STOPE.

Permanent support of roof is provided by pillars of ore flanking the stopes, the use of props within the stopes being temporary only. Under normal conditions of working two forms of pillars are employed; slope

FIG. 15.—STOPE SHOWING ARCH PILLAR.

pillars, paralleling the slopes and protecting them from caving ground in adjacent stopes, and arch pillars situated between stopes, *i. e.*, forming the upper rib of one stope and the lower rib of the stope immediately above; see Fig. 15.

With normal width of stopes of 35 ft., the pillars are 30 ft. wide and often stand unbroken for hundreds of feet; the more usual method, however, is to break them by upsets into lengths of 150 to 200 ft. When robbing is done systematically, the pillars are commonly reduced to mere stubs ranging in size from 20 by 25 ft. to 10 by 25 ft., the longer dimensions being parallel with the strike of the orebed; see Fig. 16. Through excessive use of powder and lack of care in spacing upsets, irregular sizes and shapes of pillars may result.

The gradual change in practice in handling ore by mechanical means has caused radical changes in size and arrangement of stopes. Stopes of

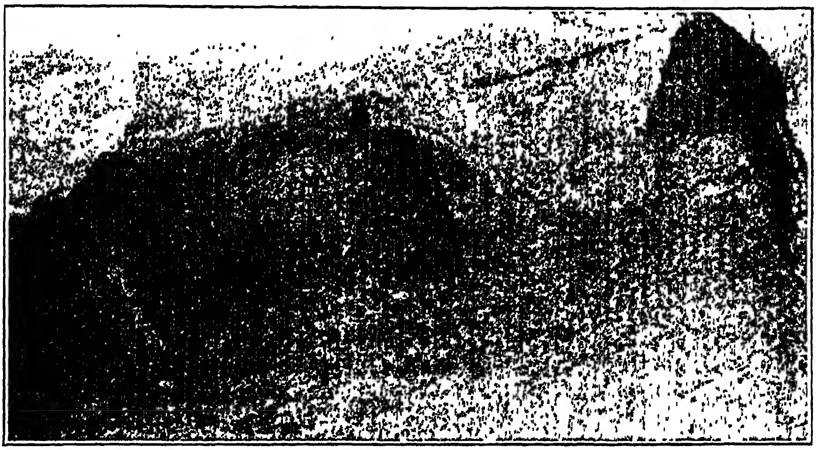


FIG. 16.—ARCH PILLAR BROKEN INTO STUBS BY UPSETS.

65 to 100 ft. wide are not uncommon and, with the use of scraper loaders, the tendency is to increase still more the size of the stopes. With the increase in area worked in stopes, the danger of falls of top rock has also increased, in spite of an extensive use of timber; this, in turn, has led to the leaving of small pillars within the stopes at such points as show weakness of roof. These wall pillars are designated as "captain's," pillars, as their location is determined by that official.

The practice now commonly followed in the larger mines is to mine to a percentage basis, 70 per cent. of the ore being removed and 30 per cent. being left as support. This means that with normal widths of stopes of 35 ft., the 30-ft. pillars are reduced by driving upsets and slabbing until 30 per cent. of the ore remains and pillars 20 ft. wide by 50 to 100 ft. long are left.

Pronounced failure of pillars often noticed in localities where extensive robbing has been done is not noticeable where mining has been done on the 70 per cent. basis. However, there are many locations in the

recent workings where pillars representing 30 per cent. of the orebed are showing signs of distress, the ore being traversed by numerous cracks.

The problems now requiring attention are threefold: (1) how to support the roof in abandoned workings; (2) how to support adequately the roof in the present workings; and (3) how to support the roof in a mining practice that will involve the extraction of both upper and lower branches of the Big Seam. The first two conditions require immediate attention, the last is a reasonable possibility and should be given careful consideration.

The support of the roof in old workings is imperative from the standpoint of protection from influx of water from water-bearing formations above the orebed. The support of roof in present workings involves such elements as high percentage extraction of ore and safety and efficiency in operation.

Can the fact that pillars are failing be taken as an index of inadequacy of support under past and present practice? If support of the roof of the stopes until such a time as the ore can be safely removed is the goal sought, support of roof may be considered as adequate, but in view of conditions involving drainage and future methods of mining, the support provided is not adequate, inasmuch as the pillars are failing extensively. A careful investigation was undertaken to determine the cause of failure, which might serve as a basis for suggestions as to abatement of the trouble or its prevention. However, as the character and condition of both ore and top rock have a direct bearing upon failure of both, they should be considered prior to a discussion of cause of failure.

In Fig. 5, the conditions affecting support of roof are shown to advantage, index numbers indicating the relative value of the elements of support throughout the active portion of the district. The principal elements considered are: percentage of slate and angle of break of top rock; thickness of sandstone and roof ore; and effect of cross-bedding on strength of top rock, given in terms of the ratio between length and depth of beam resulting from cutting of beds by cross-bedding planes.

Strength of Top Rock

The basis for judging the strength of slate and sandstone top rock as a supporting medium is the relation of span of beam to its depth, as determined from the modulus of rupture in pounds per square inch. The averages of a limited number of tests (22) give the moduli of rupture as follows: slate 1,617.08, ore and slate 2,312.25, and ore or sandstone 1,863.33.

In Fig. 17 are shown the maximum, minimum, and average depth of beams of rock that will fail under their own weight with given spans. For instance, taking the normal span of 35 ft. for roof in stopes, the

depths of beams that will fail under their own weight are as follows: slate 1.2 ft., slate and ore 0.6 ft., and ore or sandstone nearly 0.8 ft. It is evident, then, that slate and ore intimately interstratified give the most satisfactory roof material.

The results of the tests on strength of beams of top rock are better than could be expected under unusual conditions as the specimens were

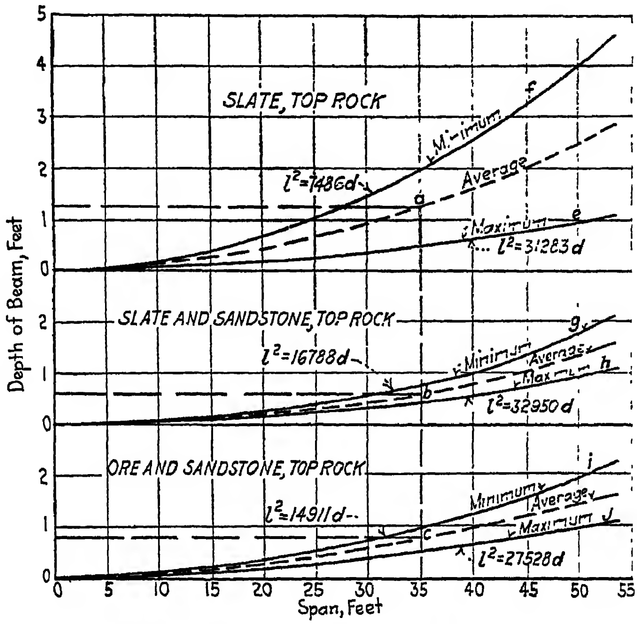


FIG. 17.—RELATION OF SPAN TO DEPTH FOR SLATE, SLATE AND SANDSTONE, ORE OR SANDSTONE.

free from slip planes and fairly uniform in thickness; however, they are relative and give a means of comparing the strength of different kinds of top rock as elements of support.

Strength of Ore

The strength of ore, while of secondary importance to that of top rock, as the latter always fails first and therefore determines the character and conditions of support, is of great importance as it determines the extent of failure of top rock, involving drainage and the ultimate integrity of the mines.

Considerable work has been done relative to the determination of the strength of ore, the object being to ascertain definitely safe working loads for pillars, the most desirable shape, and the proper position of pillars in the workings. Further, anticipating possible additional height of pillars in further mining practice, the strength of ore throughout the vertical division of the workable bed has also been investigated.

Preliminary tests of the compressive strength were made, by courtesy of the Carnegie Institute of Technology in its laboratory at Pittsburgh, on 33 cubes of four sizes cut from blocks of ore, in order to ascertain if possible the most satisfactory size. These tests gave the following results:

Number of Cubes Tested	Size of Cubes, Square Inches	Compressive Strength, Pounds per Square Inch
10	2.00	11,391
8	4.78	12,002
4	9.17	13,522
11	15.97	14,994
33	7.98	12,977 Average

The increase in compressive strength noted is due not so much to the fact that large cubes actually give an increased strength, but that the smaller cubes make less perfect contact with the plates of the testing machine, because of difficulties experienced in cutting plane faces. However, in view of the results obtained from these tests, it was decided to use cubes between 3 and 4 in. in size. Masses of iron ore were secured by the operating companies representing each foot of the bed worked from five locations in the mines on Red Mountain. From this ore, cubes were cut and ground to a hone finish. The Bureau of Standards generously coöperated and made compression tests of several kinds on the 110 specimens in its testing laboratory at Washington. The average compressive strength of 110 cubes is 13,864 lb. per sq. in., or 887 lb. per sq. in. more than the average of the four sizes in the preliminary tests. The former value is, however, more nearly representative and should be taken as the compressive strength of the iron ore throughout the vertical dimensions of the upper bench of the Big Seam on Red Mountain. Further, the 13,864 lb. per sq. in. represents the compressive strength of the ore under static conditions.

As failure of pillars appears to be most pronounced near caved ground, it seemed reasonable to assume that the intense vibration caused by falls of thousands of tons of top rock might contribute to this failure. With this in mind, a series of tests were made with an attachment to the testing machine that would permit of more or less intense vibration to be given to the test piece while under increasing pressure. The results were surprising in that an average increase of 28.3 per cent. was obtained over the tests made under static conditions. However, subsequent tests made on duplicate test pieces with the attachment in place, but without vibration, gave results 25.3 per cent. in excess of the static tests, and 3 per cent. less than the vibration tests. It is evident then that vibration was responsible for only a small part of the increase noted, but that the main cause of the increase was a more perfect contact between the plates of

the testing machine and the test pieces, a thinner and more flexible plate being employed with the vibratory attachment. The averages of the three sets of tests, which may be indicated as normal, vibratory attachment, and vibration are 11,062, 13,864, and 14,192 lb. per square inch.

The difference in the compressive strength noted between the two tests with vibratory attachment and vibration may be due to the condition of the two pieces rather than the effect of vibration, for the number of tests made in each instance showing an increase one over the other were approximately equal, the average of the tests bringing out the difference

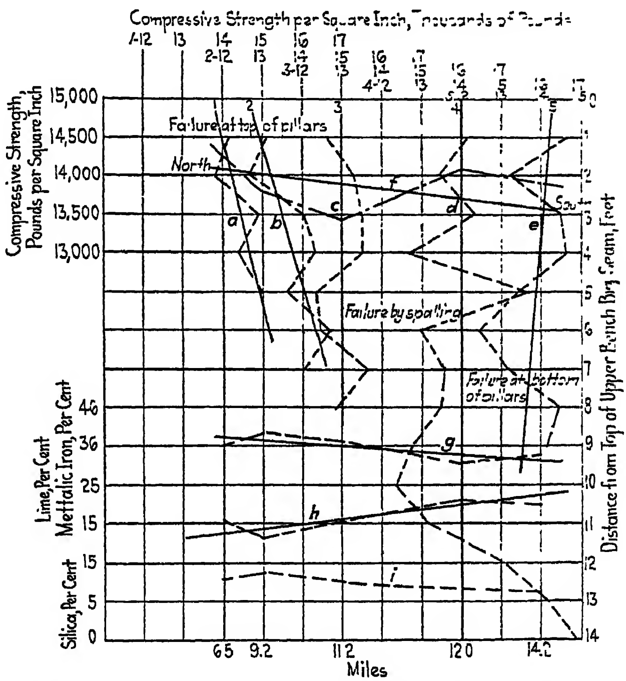


FIG. 18.—VARIATION OF COMPRESSIVE STRENGTH AT INTERVALS OF APPROXIMATELY 2 MILES, BEGINNING AT A POINT 6½ MILES SOUTH OF RED MOUNTAIN GAP, GOING SOUTH.

noted. Greater refinement in testing both with regard to equipment and method of procedure may show a still greater increase over the results obtained from the normal static tests.

An examination of the compressive strength obtained on samples of ore taken from the active mines shows a decrease from the north to the south along Red Mountain; see Fig. 18. In the same figure is given the relation existing between the constituents of the ore and the compressive strength, lines *f*, *g*, *h* and *i*, which show that the compressive strength varies directly with the iron and silica contents, while the lime content increases with decreased compressive strength. The variations of iron

and silica, while small, are significant; they give a decrease in compressive strength of 525 lb. per sq. in. for a decrease of 6.8 and 4.0 per cent. of iron and silica, respectively. As iron oxide is a bonding material and silica is not, to any considerable extent, and as they maintain relative proportions, iron must be largely responsible for the strength of the ore as measured by its compressive strength.

In Fig. 18 is also shown the variations in strength from the top to the bottom of the orebed worked at five different localities. In Nos. 1 and 2, the upper portion of the bed is weakest; in Nos. 3 and 4, the average strength is uniform from top to bottom; while in No. 5, the top of the bed is weakest.

A careful examination of the constituents of the ore for each foot of the bed worked reveals no reason for the variations observed. In No. 1, the average percentage of lime and silica increases from the top to the bottom of the bed, the iron content diminishing; in No. 2, the average of lime, iron, and silica remains constant, increases and diminishes respectively from top to bottom; in Nos. 3, 4 and 5, the average of ore constituents remains constant. It is evident, therefore, the variations in compressive strength cannot be attributed to variations in constituents.

It has been observed, however, that in the northern part of the district the tops of pillars fail first, in the central part of the district the pillars fail more or less uniformly from the top to the bottom, while in the southern part the bottoms of the pillars show pronounced failure over the middle and upper parts of the face. The natural inference would then be that it is the physical condition of the ore that is responsible for the location of failure of pillars, the presence of slips and fractures due to folding and faulting of the orebed contributing largely to such action. This thought is substantiated by the fact that the ore is weakest in that part of the district that has been subjected to the greater disturbance of strata. As previously pointed out, the orebed and top rock are badly broken by slips and cross-bedding planes, which must seriously weaken the formations and contribute materially to their disintegration and failure. With these facts well established, a definite basis is had for the intelligent consideration of support in all particulars, involving the maintenance of pillars and roof, and the integrity of the mine as a whole through prevention of extensive falls and squeezes.

Failure of Roof or Top Rock

The maintenance of the roof of mine workings is the main object of support. The ideal condition is where the top rock can be depended on to give the required support with or without resort to timbering or other forms of support either temporary or permanent. The conditions affecting failure of roof in the iron ore mines of the district are:

1. Occurrence of top rock.
2. Presence of water in bedding and slip planes.
3. Air slake acting on weakened and fractured formations.
4. Weakening of top rock by careless placing of holes and excessive use of powder.
5. Weight of cover causing bending, breaking, and shearing of top rock.

The lines of weakness as slip and cross-bedding planes are possibly the most important factors contributing to the failure of top rock. Both limit the continuity of otherwise thick and strong formations that serve as beams in the support of the overlying formations, some of which may be inherently weak.

FIG. 19.—EFFECT OF WET ROOF ON FAILURE OF TOP ROCK.

The presence of water in slips and disturbed ground adjacent to faults not only weakens the top rock but the ore in the pillars at such points. When considerable water enters the mine in this way, it usually causes the entire roof to become wet and adds materially to the effect of air slake; see Fig. 19.

The effect of air and moisture entering the roof formations as they take weight and separate along the bedding planes undoubtedly has a pronounced effect in weakening and causing disintegration of the beds, particularly the beds of slate. The softening and swelling of the slate has been observed in a number of places and demonstrates conclusively that a deteriorating action is in progress.

Top rock is often badly broken by carelessly placed shots and by powder used in excessive amounts. In those locations where roof ore is left either as temporary or permanent support, and particularly where

there are no well-defined planes of separation between the ore below or the roof ore and top rock, lack of care in placing holes may be responsible for premature and extensive falls of top rock.

Weight of Cover the Cause of Failure of Top Rock

The weight of cover, while the ultimate controlling factor in support of roof in mine workings, may be of secondary consideration if other conditions are right, such as arrangement, thickness, and strength of top rock, and of more importance, the size of the openings made in the orebed. The factors contributing in this connection to inadequate support are: varying thickness and amount of slate and sandstone in the roof and the presence of irregular formations, as pots, rolls, etc., and other irregularities. Thick, strong beds not broken by lines of weakness will, in themselves, if occurring close to the bed worked, give adequate support with properly proportioned width of openings, any thinner and weaker underlying formations requiring support for themselves alone.

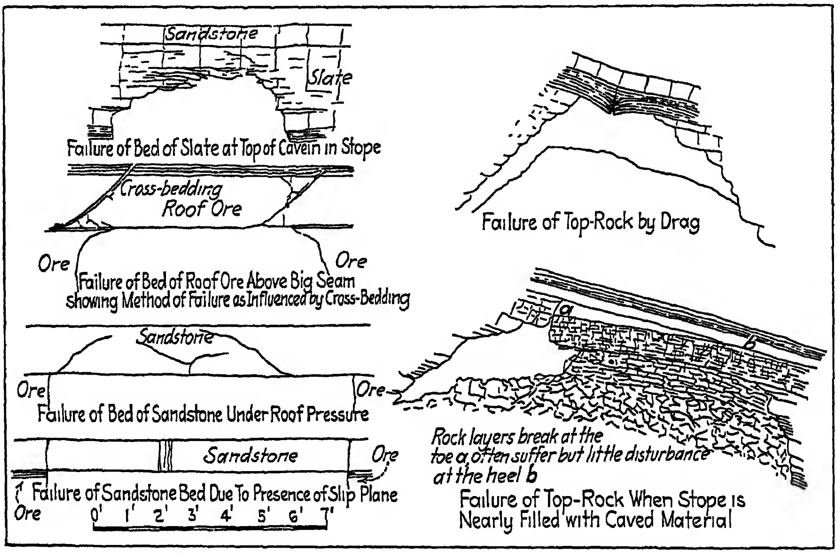


FIG. 20.—EXAMPLES OF FAILURE OF TOP ROCK IN MINE WORKINGS.

The fall of top rock in unbroken beds roofing mine workings is due to failure through rupture of the material in the lower part of the formations; while with broken beds, after the initial break has occurred the fall is due to failure of the material in the upper part of the beds. The tensile strength determines the load sustained. The presence of slips and cross-bedding changes these conditions greatly, and under certain conditions causes failure of beds by shear. Examples of failure of top rock under the conditions just noted are shown in Fig. 20.

The slate top rock, commonly known as "draw slate," as a rule is so friable that it has led to the belief that the roof exerts no pressure upon props, few props being observed that show the effect of taking weight.

FIG. 21.—PROPS STANDING IN STOPE, TOP ROCK HAVING BROKEN AWAY FROM TOP.

This is because the slate breaks around the top of the props as soon as the weight comes upon them, freeing the props and relieving them from pressure; see Fig. 21. The same condition may be observed with pillars, the slate breaking close around the tops and not infrequently breaking

FIG. 22.—TOP ROCK BROKEN TO FACE OF PILLAR.

away for some distance back of the face, which may be responsible for the tops of the pillars receding in certain instances more rapidly than the bases; see Fig. 22.

Failure of Pillars

The failure of pillars in the iron ore mines is of considerable importance in that it is indicative of more serious conditions that may be expected to follow higher extraction and the extension of the workings to greater depths. Failure of pillars is due to the following causes:

1. Effect of air and moisture in slip planes and fractures, or as commonly expressed "air slake."
2. Weakening of pillars by careless shooting, which involves careless placing of holes and excessive use of powder.
3. Pressure of cover acting through: (*a*) Top rock as downthrust or "drag," (*b*) bottom rock as "heave," (*c*) pillars as lateral expansion of face or "spalling."
4. Pressure of waste rock in caved stopes, breaking and forcing pillars out of position.
5. Movement of top rock producing a tendency to overturn the pillars.

Air Slake as a Cause of Failure of Pillars

Inasmuch as air slake was formerly commonly held to be responsible for the disintegration of pillars, an investigation was made to determine what the effect of air slake is and how pronounced the action. A location was chosen in a stope where failure was in progress and from the corner of a pillar about 5 ft. was broken, exposing fresh ore for 5 or 6 ft. along the face of the pillar. Samples were then taken every 2 in. for 60 in., beginning at the edge that had been exposed to air and moisture for a number of years. The thirty samples were analyzed for calcium carbonate, carbon dioxide, and combined water. Then to determine the effect of weathering on the iron, alumina, and silica contents of the ore, analyses were made on a limestone-free basis. The results of the analyses are given in Fig. 23, which shows that beyond a depth of 6 in. the effect of weathering is negligible, the constituents remaining practically constant. In the case of calcium carbonate and carbon dioxide, the changes noted are most pronounced; they rise abruptly to a point 6 in. from the surface and diminish from that point along the pillar. The abrupt rise noted both from the surface to the highest point *a* and at points *b* and *c* are caused by concentrations of lime along cross-bedding planes; the intervening low points mark the location of slip planes along which water has leached out the lime. While the heavy lines, in the case of carbon dioxide and carbon monoxide, represent average conditions, were it not for the high concentrations of lime, the average for these constituents would be more fairly represented by the lines *xy* and *zw*. In fact, it was a mere accident that the pillar was formed so as to show the high lime within 6 in. of the surface, which might just as well have been between *b* and *c* or any other point beyond. It is doubtful whether weathering or air

slake has affected the pillar to a greater depth than 6 in., which is evident from the soft condition of the surface ore.

As a further check on the effect of air slake as a cause for breaking of pillars, masses of ore, tons in weight, lying in stopes were observed for a period of a year or more and no disintegration was noted, while the pillars from which the masses came continued to fail. It is evident, then, that air slake as a cause of failing pillars may be dismissed as unimportant, and that the main cause of this action must be sought elsewhere. However, the leaching of the ore adjacent to slips undoubtedly has considerable effect in the weakening of the pillars.

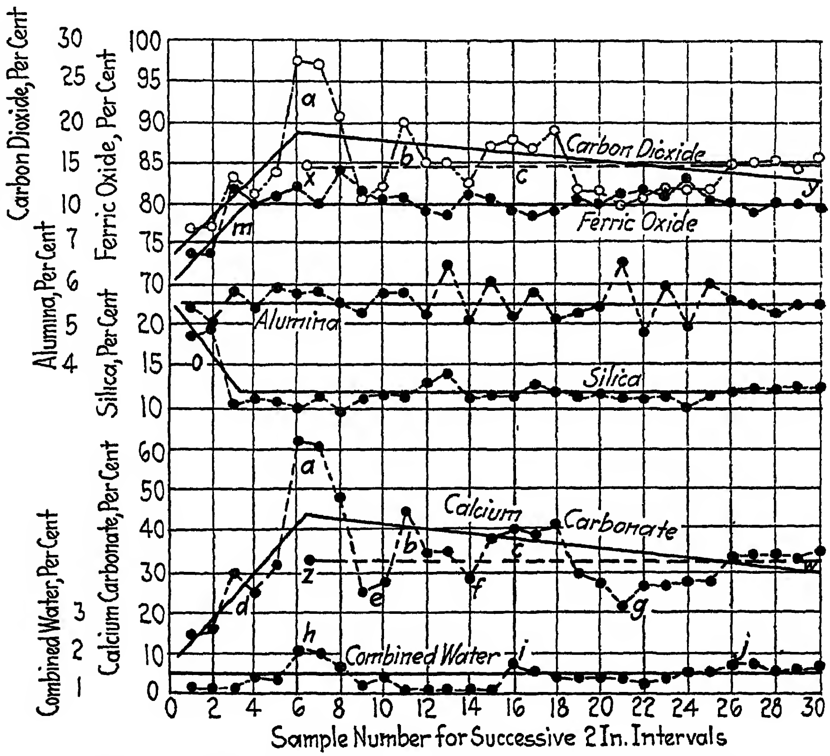


FIG. 23.—VARIATIONS OF CONSTITUENTS OF ORE FROM SURFACE OF PILLAR TO A DEPTH OF 60 INCHES.

There is no doubt but that pillars are badly broken and seriously weakened by careless shooting and excessive use of powder, but that the fresh cracks and progressive breakdown of pillars are the result of shooting done years ago needs no serious consideration. In the investigation of failing pillars, no case was considered except where evidence showed plainly that no shot had been fired in the immediate vicinity, and that the breaks were fresh.

Effect of Weight of Cover as a Cause of Failure of Pillars

A careful study of failing pillars extending throughout the old and active workings for a period of 18 months or more has shown conclusively that pressure of the cover is directly responsible for the breaking down of pillars noted.

FIG. 24.—HEAVE IN BOTTOM ROCK.

The effect of weight of cover on the bottom of stopes, acting through the pillars, is well shown in the phenomenon of "heave," the bottom rock rising in arches or breaking in well-defined lines across the stope

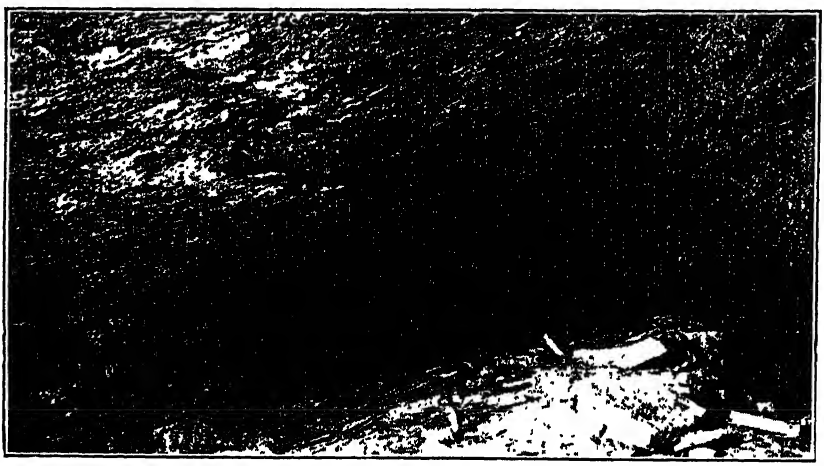


FIG. 25.—DRAG IN TOP ROCK.

bottoms. The same phenomenon as the result of "down-drag" of cover is evident in the top rock or roof of stopes, but as gravity also acts and causes the broken and loosened rock to fall, much of the evidence of

"drag" is removed at an early date; the action of drag is nevertheless definite and pronounced. As drag and heave are due to the same cause and act in a similar manner they may be considered together; see Figs. 24 and 25.

The downward bending, or drag, of the roof and the upward arching of the bottom rock results in a concentration of weight at the face of the pillars, causing them to break and disintegrate, the action continuing progressively with the recession of the face.

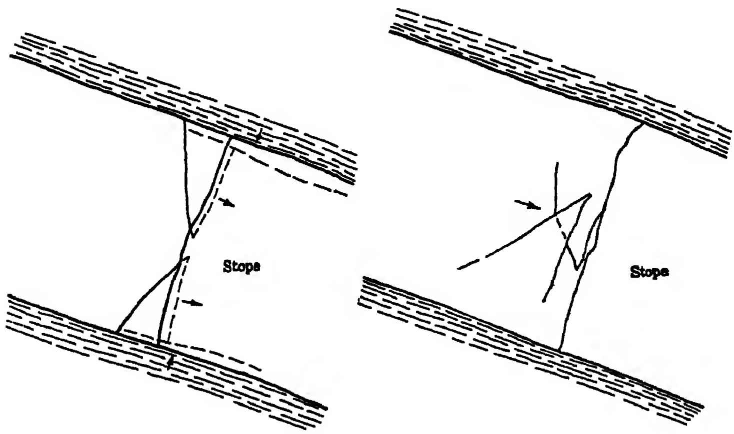
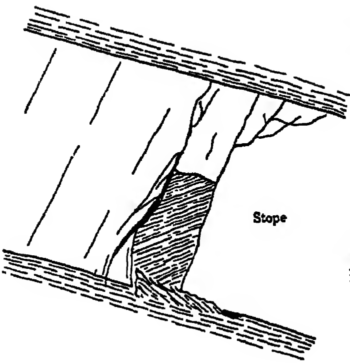


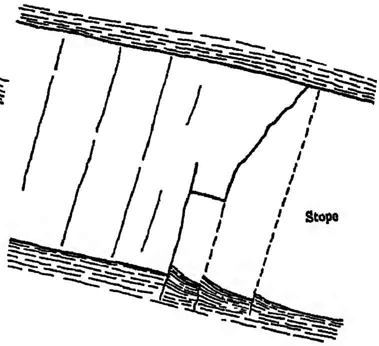
FIG. 26 (*a*).—WEDGE-SHAPED MASSES OF ORE BROKEN FROM PILLAR BY HEAVE OF TOP AND BOTTOM ROCK. (*b*) EFFECT OF DISTORTION OF PILLAR, SHOWING SLABS OF ORE BREAKING OFF.

The concentration of pressure usually occurs at varying distances from the face, depending on local conditions, but usually from a few inches to a foot or more. The presence of lines of weakness, as slip and bedding planes, undoubtedly has marked influence on the effect of drag and heave in causing the failure of pillars. Further, through the action of gravity augmenting drag and retarding, to a certain extent, the effect of heave, particularly with respect to clearing the face of broken material, the disintegrating action may, in some cases and under certain conditions, be more pronounced at the upper part of the face of pillars. This clearing action is pronounced on high dips and may be negligible on low dips, and has an important bearing upon support.

Drag and heave acting at the contact of the orebed with the top and bottom rocks shear off masses of ore from the pillars, often in the form of wedges, but also as slabs, that vary from a few inches in thickness to several feet. The line of shear then shifts forward beyond the face of the pillars, both at the top and bottom, and again loosens masses of ore. The successive and progressive action of shear due to drag and heave are shown to advantage in Figs. 26(*a*) and (*b*), 27(*a*) and (*b*), and 28(*a*).

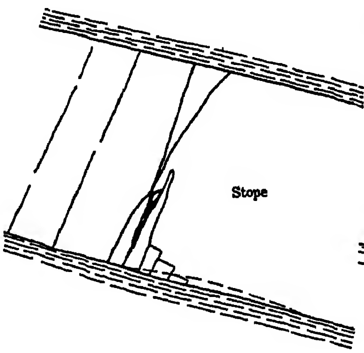


(a)

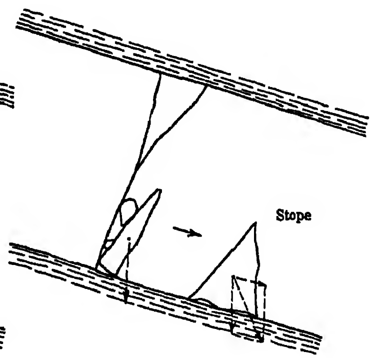


(b)

FIG. 27.—(a) MASSES OF ORE FORCED FROM PILLAR BY HEAVE OF BOTTOM ROCK. (b) HEAVE BREAKING ORE FROM FACE OF PILLARS, PARTICULARLY AT BOTTOM.

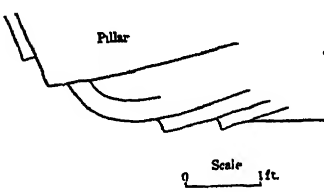


(a)

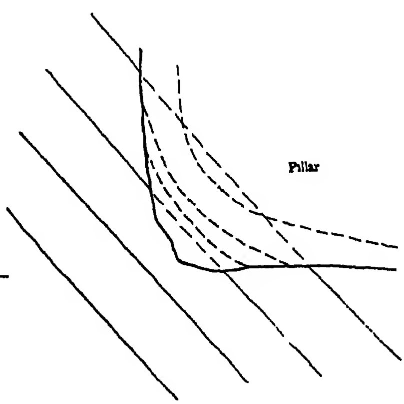


(b)

FIG. 28.—(a) HEAVE BREAKING ORE FROM FACE OF PILLARS, PARTICULARLY AT BOTTOM. (b) CLEARANCE OF BROKEN ORE FROM FACE OF PILLAR BY GRAVITY.



(a)



(b)

FIG. 29.—(a) PLAN OF CORNER OF PILLAR SHOWING ROUNDING OF CORNER. (b) PLAN OF CORNER OF PILLAR SHOWING LACK OF EFFECT OF SLIP PLANES ON FAILURE OF PILLARS.

While the action of drag and heave are influenced to a considerable extent by slip planes and cross-bedding, the evident controlling factor with respect to direction taken by the shear lines is the shape of pillars, which is shown conclusively by observation on failing pillars. Further,

FIG. 30.—HEAVE TURNING CORNER OF PILLAR.

the tendency is to eliminate all angles, as the corners of pillars, as shown in Figs. 29(*a*) and (*b*), 30 and 31.

Another important cause of failure of pillars is the spalling off of flakes and slabs from the face. This action is the result largely of the

FIG. 31.—LINES OF HEAVING ROUNDING CORNER OF PILLAR.

lateral movement or distortion of ore under pressure of cover and is particularly pronounced at points approximately midway between the top and bottom of the face; see Figs. 32(*a*) and (*b*) and 26(*b*). Drag, heave,

and spalling together cause a more or less even breaking down of the face of pillars, except where local conditions of occurrence are not favorable to their combined action.

Pillars that have been made narrow, and consequently weakened by excessive robbing or by wasting away through failure, may be completely

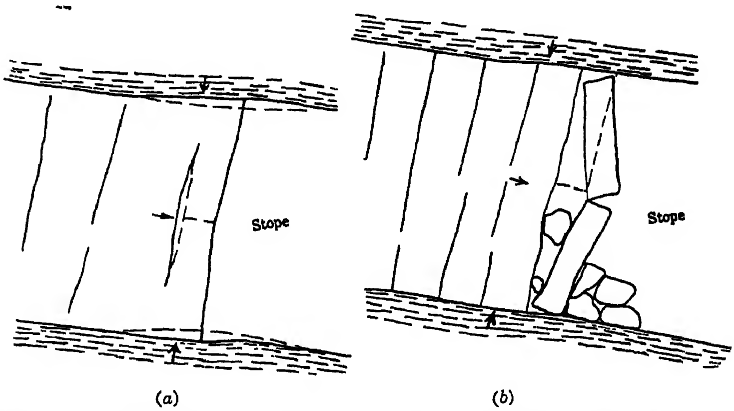


FIG. 32.—(*a*) MASSES OF ORE BROKEN FROM FACE OF PILLAR BY DISTORTION OF FACE—FIRST STAGE. (*b*) PILLAR FAILING THROUGH DISTORTION OF FACE—SECOND STAGE.

broken up by the pressure of waste rock in caved stopes. Further, the wrecking action of thrust of such caved rock may not be confined to the crushing of pillars, but may shove out of place masses of top rock, causing extensive caving. The first evidence of thrust of caved rock from the

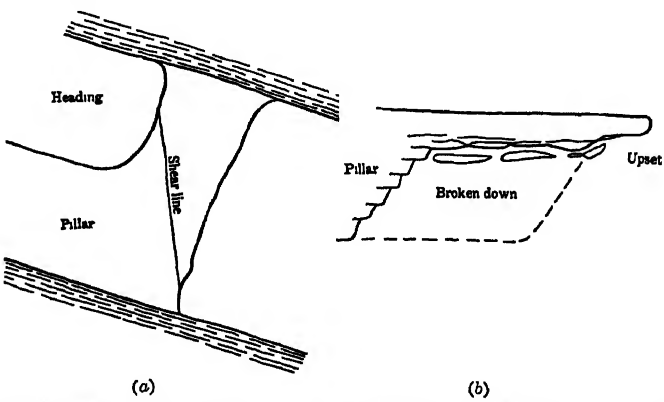


FIG. 33.—(*a*) SHEAR LINE CUTTING PILLAR SHOWING TENDENCY TO OVERTURN. (*b*) END OF PILLAR FAILING UNDER PRESSURE, PARALLEL TO DIP AND STRIKE.

stopes above may be shear lines cutting across the pillars, which is probably due in large part to a movement of roof in the direction of the dip. Shear lines may occur apart from caved ground and often where there are no evidences of movement of top rock, see Figs. 33(*a*) and 12.

Effect of Failure of Top Rock

The failure of top rock contributes, in large part, to the failure of pillars and often, being masked by certain accompanying effects, makes the main cause of failure subordinate to the effects. For instance, failing pillars are most frequently found close to caved ground, where the pillars have been relieved of great weight through falls of top rock. Obviously, reduction in load on pillars can hardly cause their breaking up, but the effect of falls on the condition of the roof might readily account for the condition observed.

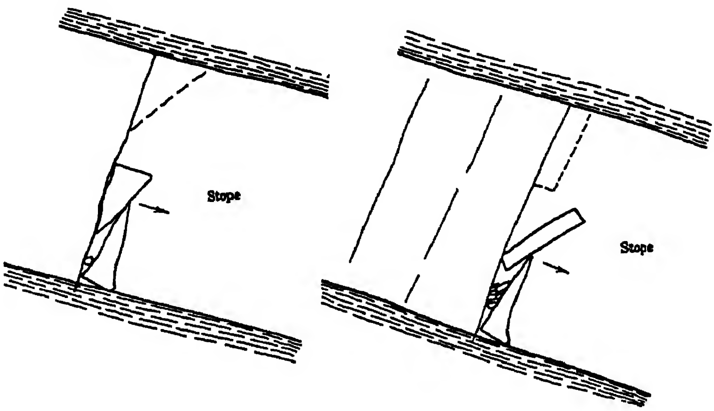


FIG. 34.—NATURAL CLEARANCE OF BROKEN ORE FROM FACE OF FAILING PILLARS.

From the time that a pillar is formed in a stope until the top rock breaks and falls, the weight of the roof is thrown more and more upon the edge next to the face, and while the pillar may not show actual disintegration until later, it has suffered severe fracturing. In addition, the fall of top rock may precede the actual failure of the pillars, but through the fall of adjacent areas of top rock they are further weakened by the drag of the roof, thus actually throwing more weight upon the face of adjacent pillars.

The fall of top rock, and consequent weakening of extended areas of roof, is, therefore, progressive until considerable areas of stopes are filled, or partially filled, with caved rock and a corresponding amount of pillar has been weakened if not actually broken. Once begun, the failure of pillars continues both through direct pressure of the roof and the concentration of weight at or next to the free face of the pillars.

When beds of sandstone occur directly above the orebed, or close to it, the effect of drag is most pronounced, as the beds do not break close to the pillars but at some distance from them. When slate occurs immediately above the orebed, the top rock may break close to the pillars, thus tending to shift the weight of the cover inward from the face of the pillars although the load will still be concentrated largely near the face.

While the failure of pillars takes place both on the upper and lower ribs of the arch pillars, by far the most pronounced action is on the lower ribs, or at the top of the stopes. The most satisfactory explanation for the greatly reduced failure of the upper ribs, compared with the lower ribs of pillars, is that owing to the dip of the formation the caved rock fills the lower part of the stopes and, by supporting the sides of the pillars, prevents their disintegration. Besides, the thrust of the top rock against the heel of the stopes prevents the ready breaking and shearing so common at the toe of the stopes; the pressure holds the top rocks in place, while tending to pull them apart at the top of the stopes.

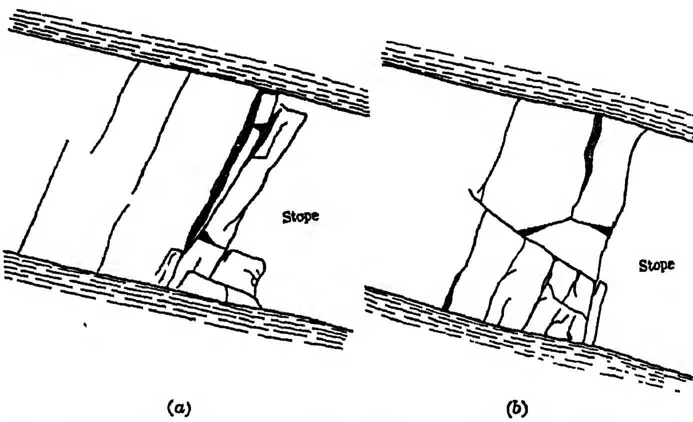


FIG. 35.—(*a*) PILLAR FAILING UNDER DISTORTION OF ORE AND HEAVE OF BOTTOM ROCK. (*b*) PILLAR FRACTURED BY ROOF PRESSURE AND HEAVE.

The observations on failing pillars made throughout a wide extent of mine workings are corroborated by compression tests on cubes cut from masses of ore taken from the same mines, also by tests made on crystals and rocks under pressure.[1]

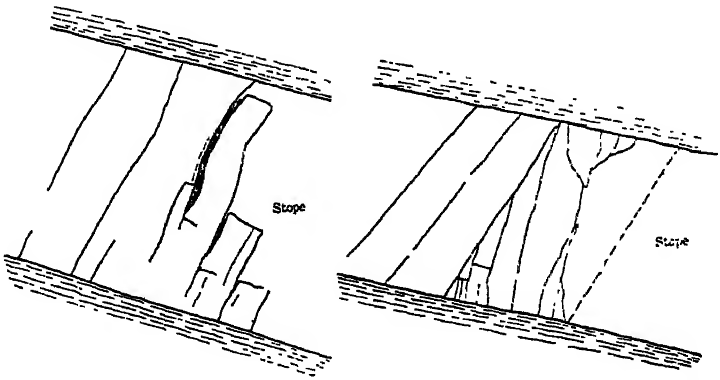
The results of the various tests, which were similar in effect, may be stated as follows:

1. Pressure causes slivers to break off from surfaces of contact, because of irregular contact with fractures and bedding planes, Fig. 42 (*a*).

2. Lines of weakness, as cleavage or slip planes, do not control the manner or amount of rupture, but have considerable influence when the surface coincides with the direction of such line of weakness, Figs. 32 (*b*) and 35 (*a*).

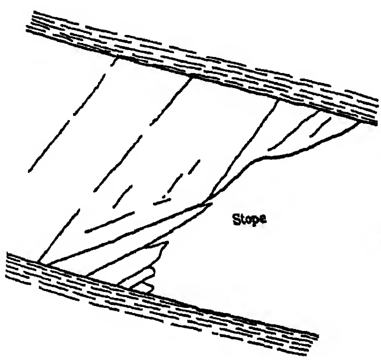
3. Cleavage or slip planes may be responsible for failure in relatively large masses through movement along such planes.

[1] P. W. Bridgman: Failure of Cavities in Crystals and Rocks under Pressure. *Am. Jnl. of Sci.*, 4th series, **45**, 243.



(a) (b)

FIG. 36.—(a) FAILURE OF PILLAR BY DISTORTION AND HEAVE. (b) EXAMPLE OF FAILURE OF PILLAR.



(a) (b)

FIG. 37.—(a) UNDERCUTTING OF FACE OF PILLAR BY HEAVE. (b) UNDERCUTTING OF PILLAR BY FAILURE THROUGH FAILURE BY HEAVE.

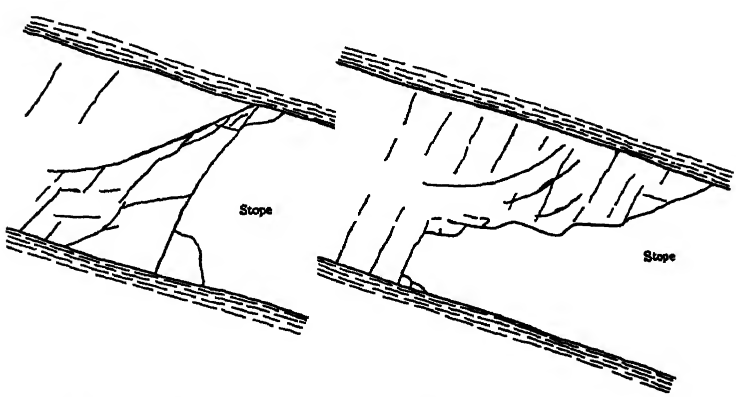


FIG. 38.—EXAMPLES OF UNDERCUTTING OF FACE OF PILLAR BY HEAVE OF BOTTOM ROCK.

4. Failure is caused largely by scaling or spalling from the face under pressure, the resulting surface being smooth in small and rougher in large masses.

5. Reentrant angles or cavities formed at one point, as the result of local conditions, may be the seat of excessive erosion or disintegration, Figs. 37 (*a*) and (*b*) and 38.

6. Surface fractures are not always indicative of failure within the mass.

Examples of Failure of Pillars

Several hundred locations in abandoned stopes, as well as other points in the more recent workings where pillars are failing, have been

FIG. 39.—PILLAR FAILING UNDER PRESSURE OF TOP ROCK.

examined and a careful study made of conditions affecting failure of both top rock and pillars. The illustrations of failing pillars, shown in Figs. 33 (*b*) and 35 (*a*) to 40 give an excellent idea of the character and extent of failure caused by pressure of cover. It is interesting to note that the failure forms obtained in testing cubes are occasionally observed in the mines, particularly in the case of small pillars; see Figs. 42 (*b*), 43 and 44.

Clearance of Broken Ore from Face of Pillars

The failure of pillars may be hastened or retarded by the promptness with which the broken ore is removed from the face of the pillars. On high dips, the clearance of ore from the foot of pillars takes place at once, provided it is not obstructed by fallen ore or top rock or by props; on low dips, the broken ore may pile up against the face and may contribute largely to support by holding the fractured ore in the pillars in place.

Further, the size and shape of masses of ore broken from the pillars control, to a large extent, the movement of ore from the face. Thin wedges will overturn and, unless the dip is sufficient, will remain close to

FIG. 40.—WEDGE-SHAPED MASSES OF ORE BREAKING FROM FOOT OF PILLAR.

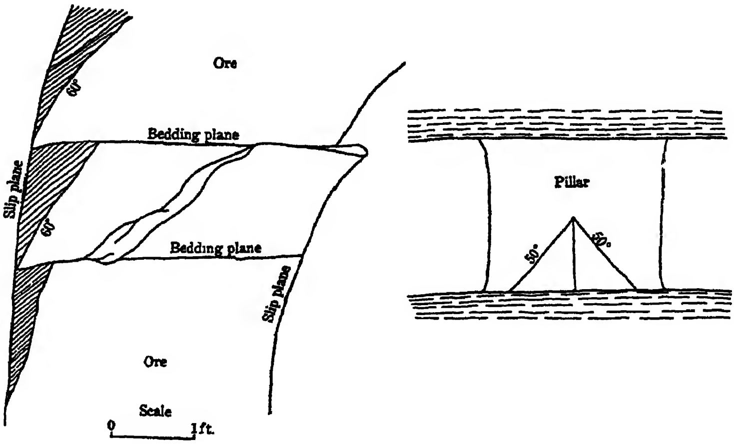
the face; while if wedges with wide bases are formed, they tend to slide and, if heavy, will travel some distance; see Figs. 29 (*b*) and 45. The material broken from the top of the face of pillars always falls free from

FIG. 41.—PILLAR COLLAPSING IN ROOF PRESSURE.

the pillar, unless prevented from moving from the foot of the pillar by accumulations at that point. The natural breaking and clearance of loosened ore from the face of a pillar are shown in Fig. 34.

Results of Study of Roof Support

The investigation of top rock and ore and conditions affecting them has established some facts regarding their strength that have a definite



(a) (b)

Fig. 42.—(a) Portion of side of pillar showing bracketing of ore between bedding and slip planes, heave acting at bedding planes. (b) Pyramid formed at base of pillar failing under pressure.

bearing on the support of the roof. The problem cannot be solved in an intelligent manner without a thorough understanding of cause and effect, and an adequate measure of values of the elements involved. This is

Fig. 43.—Formation of inverted cone by failure of pillar.

particularly true of roof support, where the relations existing between formations of varying character, thickness, and strength are the con-

FIG. 44.—FORMATION OF PYRAMID BY FAILURE OF PILLAR.

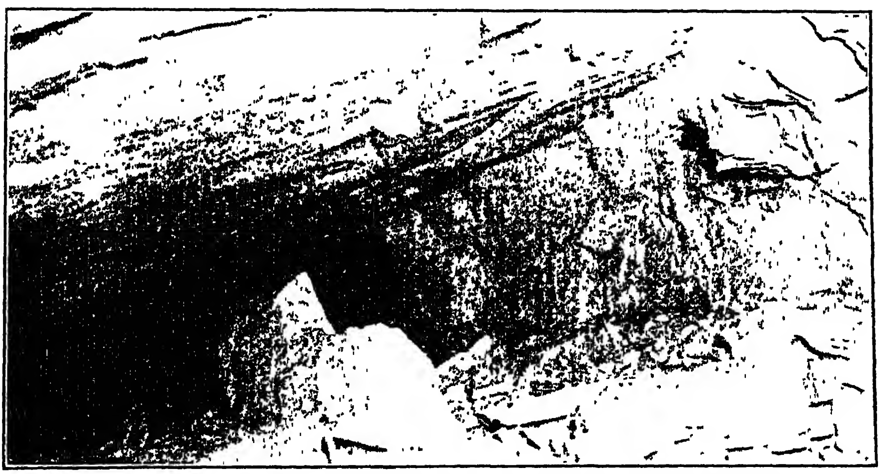


FIG. 45.—WEDGE OF ORE BROKEN FROM FACE OF PILLAR.

trolling factors. The unit values of the top rocks acting as beams and the compressive strength of ore acting as supports are essential to a thorough and complete understanding of the problem of roof support.

Top Rock as an Element of Support

The formations overlying the Big Seam vary widely in character and occurrence throughout the district. As a rule the slate is very friable, breaking readily over props and along the face of pillars, rendering support by props temporary at best. However, in certain localities the slate is tough and strong and stands well, even for long periods. The difference noted is due, however, more to thickness of bed than to character of material; the thinly stratified beds are exceedingly weak and often break and fall under their own weight for moderate spans.

The occurrence of roof ore makes it possible to support, with a fair degree of safety, what would otherwise be very weak roofs. This ore is variable in thickness but is usually sufficiently thick to support the weak draw slate immediately above and to prevent air and moisture coming in contact with the slate.

Probably the best roof is composed of ferruginous sandy slate, which not infrequently occurs at the top of the orebed. This fact is shown in Fig. 17, where the lengths or spans of beams of any thickness that will fail under their own weight are given. A thickness of 0.6 ft. of slate and sandstone will break with a span of 35 ft , compared with a thickness of nearly 0.8 ft., and 1.2 ft. for ore or sandstone and slate of equal span. While the difference is not pronounced in the case of ore or sandstone compared with slate and sandstone, it is pronounced in the case of slate alone.

Any condition that disturbs the beds through breaking the continuity of the beams and loosening the bond between the respective layers of top rock accentuates a bad condition. The presence of slip planes, cross-bedding, folds, and faults are the main contributing factors in weakening of top rock. Means that suggest themselves as offering some measure of relief in the support of roofs are: (1) The use of an adequate number of properly placed props, particularly in wide portions of stopes; (2) the forming of pillars so as to prevent long exposures of slip planes in the top rock; (3) the use of comparatively narrow openings in the preliminary mining or the advancing work; (4) the prevention of extensive caving of top rock and failure of pillars in adjacent workings which disturb and throw additional weight upon the roof of working places; and (5) causing the roof to cave in worked-out stopes and by filling the stopes with caved rock prevent further falls of top rock. A study of occurrence of top rock as exposed by falls will prove useful in determining the span of beams, and consequently the width of working places.

Ore as a Means of Support

The support of roof by pillars of ore, while affected and modified by the character and strength of top rock, has mainly to do with depth of cover overlying the bed worked. The weight of cover rests upon the pillars and must be supported by them irrespective of the size and strength of the beds composing the top formations.

When considering the strength of pillars, the entire weight of cover must be considered, the reduction of such weight by arching, or "overhang," from unmined areas cannot be considered, for a release at one point means a concentration at another point, the average remaining the same, and is the controlling factor in determining the support to be provided.

The thickness of cover on the orebed, for area worked, is well under 1000 ft., while the maximum thickness of cover for the same area is under 1500 ft. Assuming a weight of 1 ton for each 13 cu. ft. of top rock, which is a fair average, there would be a load of 15.55 lb. per sq. in. at the base of a column of rock 1 ft. square by 13 ft. high, or for each 100 ft. of cover the load per square inch would be 119.58 pounds.

Under conditions existing in mining where support of roof is necessary because of removal of the ore, added weight is thrown upon the unmined portion in the bed. If one-half of the bed is removed, the weight upon the unmined portions is doubled; while if 70 per cent. of the bed is removed, the load thrown upon the portion remaining is increased by $2\frac{1}{3}$ times, or 278.62 lb. per sq. in. Assuming that mining is done at a depth of 1500 ft., in the slope mines of Red Mountain, the load due to weight of cover will not exceed 1800 lb. per sq. in., or if 70 per cent. of the ore has been removed by mining operations, the weight upon the pillars will be 4179.30 lb. per square inch.

With an average crushing strength of ore of 13,864, or approximately, 14,000 lb. per sq. in., taking the highest compressive strength obtained, the pillars would have a factor of safety of 3.3, which is inadequate in view of the weakened condition of the ore in the pillars because of slip planes, etc. That the factor of safety is much too small is also shown by a comparison with the factors employed with similar materials used in buildings where the factor of safety is 20 and above. There is little doubt but that the small factor of safety shown in a solid mass of ore is largely eliminated with pillars weakened by slip planes and by the excessive use of powder.

As the dip of the orebed is quite variable, it is not possible to state definitely how conditions of support, particularly with respect to increased weight on the pillars, will be affected with the prolongation of the slopes, as the larger number of the slopes show that the dip of the orebed is becoming less with distance from the outcrop. However, abrupt, and

often pronounced, increases in cover occur when faults are encountered and other less pronounced increases arise from hills occurring in the valley above. In some instances, the increases in cover noted are compensated for by anticlinal folds in the orebed itself, but as a rule the cover is increasing in depth in a more or less regular way.

The increase in cover resulting from faults may amount to several hundred feet; but with land elevations, nothing to exceed 100 ft. is encountered until Shades Mountain is reached, which is too far away to be seriously considered at present. The first pronounced change in elevation of the surface is a low ridge that parallels Red Mountain from Woodward No. 1 mine northward to the neighborhood of Ishkooda and lies to the eastward at a distance of 0.25 to 0.40 mile and averaging 0.31 mile.

It is reasonable to assume, then, that while there will be a gradual increase in weight of cover, there will be no abrupt variations of any consequence and the problem of support will require increased attention with the prolongation of the slopes.

In view of the information available relative to the strength of pillars and the cause and effect of their failure, the following method of procedure seems desirable when applicable: (1) Use as large pillars as possible, which suggests a caving system with preliminary openings driven narrow: (2) square pillars with rounded corners are to be preferred as they fail less rapidly than rectangular forms; (3) faces of pillars should, if possible, be made to form fairly wide angles with the direction of slip planes; (4) pillars should be offset in order to break the continuity of slip planes along which heave and drag act; (5) give adequate support by props and small pillars or other temporary support at the wider portions of workings; (6) in order to insure against collapse of pillars in worked-out stopes with weak roof, reenforce by caving top rock against them; (7) with the strength of ore known, determine safe working loads for pillars with ample factors of safety to insure permanent support.

The support of the iron ore mines is not an isolated problem but is intimately related to the development and working of the mines. Roof support in the ore mines of the Birmingham district is further complicated by a water problem which, in turn, is dependent largely on the failure of top formations or subsidence. A careful and detailed study must be made of all elements entering into and affecting support if a satisfactory solution is to be obtained.

Acknowledgments

Acknowledgments are hereby made to the operators of the iron ore mines for their coöperation in the investigations that have made the preparation of this paper possible. Particular acknowledgment is

due to C. E. Abbott, General Manager of the Tennessee Coal, Iron, & Railroad Co.; C. E. Bowron, Chief Engineer of the Gulf States Steel Co.; W. M. Lacey, former General Superintendent of the Woodward Iron Co.; W. J. Penhallegon, General Superintendent of the Republic Iron & Steel Co.; J. E. Strong, Vice-President and Chief Engineer of the Alabama Co.; H. J. Thomas, General Superintendent of the Sloss-Sheffield Iron & Steel Co.; as well as many others whose advice and suggestions have been invaluable.

Mining Methods in the Mineville (N. Y.) District

By Earl C. Henry,* Mineville, N. Y.

(San Francisco Meeting, September, 1922)

Magnetic iron ore was mined in Essex County, N. Y., during the American Revolution; Benedict Arnold is said to have mined ore near Port Henry to secure iron for chains and spikes for the Lake Champlain fleet. Details regarding the operations prior to 1804 are not easily obtained but work on a commercial scale appears to have been started about 1838. The Witherbee, Sherman & Co. was established in 1849. The property of the district is held in fee almost entirely by the mining companies.

The Mineville iron-ore district, as well as the entire Adirondack region, belongs to the pre-Cambrian complex. The readily identified rocks consist of gabbro, anorthosite, syenite, and granite, and are classed as igneous, eruptive, and intrusive. In addition, there is a great variety of gneisses, rocks with a variable mineral composition but all more or less distinctly foliated. The Grenville series are the oldest rocks in the district; this series consists of graphitic white limestones, graphitic sandstones, and quartzites and gneisses with or without graphite.

The orebodies occur in the gneissic rocks and are comformable to the structure of the rocks. The typical occurrence of the ore in this district is in lenticular bodies with a parallel pitch to the southwest; this structure is best exemplified in the New Bed mine. The Harmony and Old Bed orebodies are faulted by two sets of diabase dikes, one set having a southwest strike and the other set an east-west strike. The ores occurring in the acid gneissic rocks are non-titaniferous. The ores occurring in the gabbros and anorthosites are high in titanium; these deposits are irregular, have no structural relation with one another, and are unquestionably of igneous origin. The origin of the gneissic rocks and the ores therein contained is not settled; both the sedimentary theory of ore concentration and the igneous theory of magmatic segregation have been advanced.

The iron mineral in all the orebodies is crystalline magnetite; some very perfect and large specimens have been obtained. The gangue is

* Witherbee, Sherman & Co.

siliceous in all the ores and the phosphorus varies from 1.50 to 0.04 per cent. in the crude in the different ores. In the Old Bed mine, the gradation from pure ore to wall rock is sharp; while in the Harmony and New Bed mines, the gradation is gradual. The district is overlain with glacial drift. The Harmony orebody does not outcrop; it was found by magnetic needle observations.

Mineville is situated 6 miles northwest of Port Henry, which is the shipping point for the ores, by rail on the Delaware & Hudson railroad or by water through New York state barge canal and down the Hudson River. Lake Champlain is 100 ft. above sea level and is referred to as datum. The ground rises steeply to the 500-ft. contour and this extends upward in a broad valley to Mineville; the mines are about on the 1300-ft. contour. The general valley is closed, except for this southeastern entrance, by the Adirondack foot hills on the west and north, Bald Peak and spurs on the east, and Bulwagga Mountain on the south. The Lake Champlain & Moriah R. R. serves the mines and in 6 miles of track gains 1200 ft. in elevation. The climate is similar to other districts in the northern states, with a moderate rainfall. The water supply is ample for all purposes and some water-power stations are located in the district. Lumbering has been an important industry. The lumber and some of the timber used in the operations are supplied locally; coal is shipped into the district.

The labor supply is made up of Americans, Poles, Italians, Spaniards, Hungarians, and Austrians. They are efficient mine workmen. The district is not unionized.

The diamond drill is the only machine used by the Witherbee, Sherman & Co. in exploring outcrops of unknown extent, to determine the extent of an operating orebody, or to feel out an orebody in advance of a proposed slope.

Magnetic iron orebodies of the non-titaniferous type are fairly regular. They are strictly conformable to the schistosity or gneissic structure of their hosts; formal spacing of diamond-drill holes is thus a matter of secondary importance. Dip, strike, and pitch are the controlling factors. Care is taken to locate a drill hole well within the side lines, which have been fairly well determined in the operated mine, and strictly on the strike line. So well established is the lenticular character of the greater number of magnetic iron mines that mathematical spacing by means of coördinates would lead to disastrous conclusions; the greater number of holes located on this principle would be barren, as even a large lens or series of lenses might be cut only once or twice in a dozen holes.

Determining Size and Estimating Value of Orebodies

Witherbee, Sherman & Co. has stressed the importance of core recovery; even in rock, 100 per cent. core is insisted upon within reason.

As a result, 100 per cent. core in the ore horizon is the rule. The ore horizon is divided, by eye, into rich, lean, and very lean sections; each section of the core is split and the iron and phosphorus percentages determined. The completed analysis section may then read, for example,

1.5 ft., iron 47.5 per cent., phosphorus 0.55 per cent.
3.0 ft., iron 35.0 per cent., phosphorus 0.70 per cent.
2.0 ft., iron 20.0 per cent., phosphorus 0.48 per cent.
10.0 ft., iron 45.0 per cent., phosphorus 0.50 per cent.
———
16.6 ft.

The general average, on a foot basis, is 16.6 ft. ore with an average iron content of 40.4 per cent.

To provide against possible loss of core, sludge samples have been taken. Experience shows that the sludge runs about five units in excess of iron in core; for example, the sludge from the foregoing would show about 45 per cent. iron against 40.4 per cent. shown by the core. The entire core, including the split ore, is carefully stored in racks in a core house.

The method of estimating tonnage as the result of diamond drilling is comparatively simple in the orebodies of this company. The horizontal working face of one mine, at right angles to the axis, is 1850 ft. The average thickness of the ore and the percentage of iron is determined. Three drill holes were located at an average distance of 738 ft. in advance of working faces. The average thickness of the working face and of the ore shown in the drill holes is the average thickness. This multiplied by the horizontal area gives the cubic feet of ore. Dip is disregarded, or allowed for possible errors. The tonnage will, of course, vary with the percentage of iron; if the cores show no diminution in thickness from working face, 500 or 1000 ft. in advance of drilling is reckoned as probable ore.

Mining Methods Employed

Witherbee, Sherman & Co. operate the following underground mines: Old Bed, Harmony, New Bed, and Smith. The ore is magnetite, averaging from 30 to 50 per cent. iron and varying in phosphorus. The hanging wall is hard granitic gneiss and the foot wall is hornblende gneiss. The orebodies are inclined, dipping to the southwest at an angle varying from 10° to 60°. The ore varies and is mined in thicknesses from 4 to 70 ft. Underhand stoping is used and pillars of ore are left to support the roof. No attempt is made to sort out the rock. All the broken material is sent to the mill. Timber is used only for tracks, stairways, and pocket faces.

The Old Bed, Harmony and Smith mine workings are reached by a shaft while the New Bed is served by a tunnel. Each mine has a main hoisting slope, equipped with a large storage pocket at the bottom, in which all the ore in the mine is collected. From this pocket a haulage road is driven in the foot wall and pockets made where necessary to

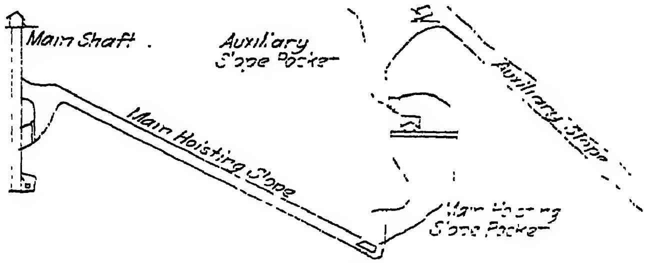


FIG. 1.—IDEAL SECTION SHOWING RELATION BETWEEN UNITS.

serve the auxiliary slope. The orebodies are faulted by dikes, so the auxiliary slopes are placed in the most advantageous locations to serve the faulted sections of the orebody. The main hoisting slope delivers the ore to a large pocket at the foot of the shaft, from which it is hoisted to the surface and delivered to the mill bin. At the New Bed mine, the

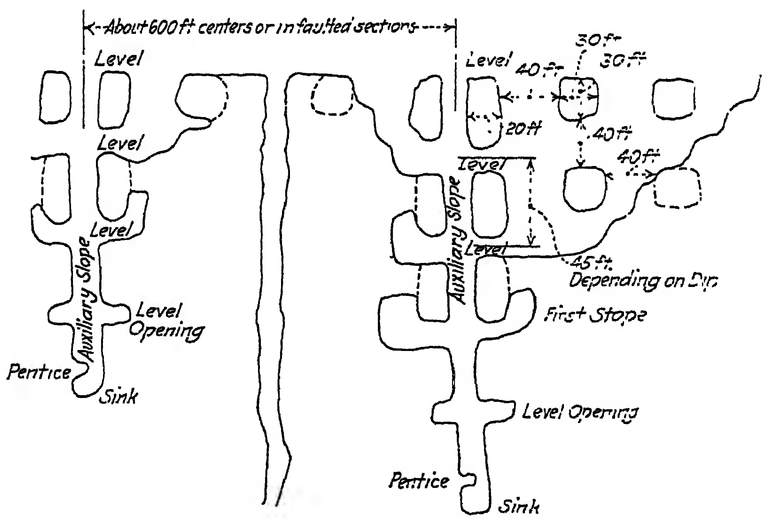


FIG. 2.—SKETCH OF METHOD OF MINING.

main slope delivers the ore to a bin on the main tunnel level, whence it is transferred to the mill bin by electric locomotives.

The drilling and blasting in each slope is supervised by a slope boss. Drilling is done by one-man hammer drills of both the wet and dry types. The light hammer drills are used in stoping and are run dry in the well-ventilated stopes, ⅞-in. hexagonal hollow drill steel being used. The

heavier water hammer drills are used in the drifts, sinks and small headings, 1¼-in. round hollow steel being used. Cross bits are used on all drills. In the drifts, the wedge or V cut round is used. In the stopes, a bench is advanced in the heading and the ore is stoped by the underhand method from this bench; 25 per cent. gelatin powder is used for both blasting and block holing.

Mine sampling, in the strict sense of the word, is seldom done. The crude ore, sent to the mill, is sampled daily by passing a pan every half hour through the stream of ore after it has been crushed to 4 in. Inspection of working faces gives a close approximation of the grade.

In the Harmony mine, six Thew shovels, type O, are being used to good advantage; three Myers-Whaley shovels are operated in the Old Bed mine.

Trolley-type electric haulage motors are used in all the mines. The main haulage roads are equipped with 7-ton locomotives having two G.E. HM819 motors and 3-ton locomotives having two G.E. HM826 motors. The short hauls and tramming from the electric shovels is done by a locomotive built in the company's shops and equipped with a G.E. 96 motor. A 54-cu. ft. gable-bottom car is used throughout. The haulage tracks are 36-in. gage and are laid with 45-lb. rails. The hand tram cars are of the end dump type and have a capacity of 1½ tons; these tracks are 36-in. gage and laid with 30-lb. rails.

Underground storage pockets of from 500 to 1500 tons capacity are provided at the head of each auxiliary slope and at the foot of the main hoisting slope. Skips operating in the auxiliary slopes dump the ore into the pocket, from which it is drawn into gable-bottom cars and transported to the main hoisting-slope pocket. From this pocket, the ore is drawn off into a measuring chute, holding a skip load, and delivered to the main slope skip by gates operated by air lifts.

The surface hoists are manufactured by Wellman-Seaver-Morgan Co.; they are of the double-drum, double-clutch and brake type and are equipped with 500-hp. motors. The main slope hoists underground are 600-hp. Nordberg design with double drum, double clutch, and brake and are equipped with two 300-hp. motors. The auxiliary slopes are equipped with Nordberg 300-hp. double-drum, double-clutch, and brake type hoists with two 150-hp. motors. These slopes are also equipped with 52 and 75-hp. Lidgerwood single-drum clutched hoists. The power supplied is twenty-five cycle, three phase, 440 volt. The electric repeat-signal system is used. The surface cages have a capacity of 20 men and the underground cages, 28 men. The cage in the vertical shaft is equipped with safety catches and the Lilly safety controller is used on the hoists operating the man cages. The surface skips have a capacity of 3½ tons and the main slope skips, 7 tons. The rope is

$1\frac{1}{8}$-in. plow steel, regular lay. The surface and main hoisting slope skips are operated in balance and the auxiliary slope skips out of balance. The hoisting speeds are: For surface hoists 800 ft. per min., main slope hoists 1200 ft. per min., and auxiliary slope hoists 600 ft. per min.

Electric pumps are used in all the mines with air pumps held in reserve. The capacities vary from 100 to 400 gal. per min. The mines are comparatively dry.

The following types of air compressors are in use:

	Capacity, Cu. Ft.	Operated By
Nordberg	5600	Steam
Nordberg	2500	Electricity
Ingersoll-Rand	2500	Electricity
Ingersoll-Rand	1200	Electricity
Ingersoll-Rand	600	Electricity
Ingersoll-Rand	300	Electricity

The ventilation is natural. The mine workings are connected to the outcrop pits and the circulation of the air is controlled by drifts and doors.

The mine is lighted by 110-volt, a. c. lighting circuit; the main stations are illuminated by flood lighting. The individual miners use carbide lamps.

A Stromberg-Carlson telephone system connects the various slopes and underground hoist stations to the mine central. The mine central is connected to the surface telephone system.

The following records of production are the results obtained for year 1921, operating at 60 per cent. of capacity. The long ton of 2240 lb. is used. All labor is employed by the day, except in stoping, drifting, and tramming; the stoping contract is figured by the ton, the drifting contract by the foot advance, and the tramming contract by the car.

	Tons Per Man Per Hour	Man-hours Per Ton
Stoping	4.6	0.217
Development	3.3	0.303
All underground labor	0.74	1.35
Total organization	0.43	2.33

The classification of labor for the total organization is as follows:

	Per Cent.		Per Cent.
Mines	58.1	Surface construction	3.5
Mills	13.4	Laboratory and testing plant	0.6
Electrical repairmen	1.3	Engineering	1.0
Foundry and shops	6.7	Office	3.0
Power plants	4.2	Hospital	0.6
Supply department	1.2	Watchmen	0.1
General surface	6.3		

The labor turnover for the year was 130 per cent.; this, however, does not mean that all new men were employed; it includes the men who drew

their time and were away for over three days before returning to work. The total labor costs are 58 per cent. of the total cost of production.

The following results were obtained from explosives during the year:

Operating	1.40 tons ore per pound of powder
Development	0.57 tons ore per pound of powder
Total operating and developing	1.10 tons ore per pound of powder

To place the ore in the head house requires 9.5 kw.-hr., which is distributed as follows: Mining (compressed air) 2.84 kw.-hr.; haulage, 0.56 kw.-hr.; hoisting, 5.30 kw.-hr.; pumping, 0.40 kw.-hr.; lighting, 0.40 kw.-hr.

The total cost of supplies used, expressed in per cent. of the total, is:

	Per Cent.		Per Cent.
Explosives	31.0	Lumber	4.6
Oils	0.6	Tools	0.7
Pack and waste	0.2	Electrical supplies	5.0
Power	24.6	Air hose	0.2
Iron and steel	9.8	Wire rope	2.6
Castings	3.1	Miscellaneous	8.9
Bolts, nuts and washers	5.9	Shop supplies	1.5
Pipe and fittings	0.9		

Supplies cost 42 per cent. of the total cost of production.

Safety and Welfare Measures

The safety program is organized through the general safety-first committee, which is composed of the heads of departments with the general manager acting as chairman. Each month, one of these members and men selected from the several departments act as a surface safety first committee and inspect the plant. The mines safety-first committee is organized in a similar manner; both of these committees report to the general safety-first committee. A prize is offered to the best safety-first suggestion proposed by an employee during the month and a reward is made for all suggestions accepted.

The welfare work is carried on by the community committee, which consists of one representative from each of the two companies operating in this district and one for the community. This committee employs a director, who has charge of all welfare work. The activities are numerous; they consist, in part, of an entertainment program at the community building, band organizations, visiting nursing, sanitation, garbage disposal, sidewalk improvements, Fourth of July and Labor Day celebrations, community fair, and athletic activities. The director and community committee also take an active interest in all matters pertaining to the welfare of the employees and the community.

Timbered Stopes

The term "timbered stope" is here meant to denote stopes in which timbering is the predominant feature of the mining method. Stopes with stull sets, as in the Hecla mine, are types of timbered stopes; these types are probably the best-known.

Timbering stopes by means of square sets is not as common as formerly, as the system is not often necessary, yet there are good examples and large tonnages are being mined by the method. The square set may be utilized in ordinary overhand stopes if the orebody is suitable. At some mines, square sets have been abandoned in favor of top-slicing; and at others, the square set has superseded horizontal cut and backfilling. A drift, either in the wall or orebody, is necessary before opening the sill floor. In wide deposits, there may be a grillage of drifts and crosscuts, and at least one raise to the level above is needed in each stope for ventilation and handling timbers and filling. The square sets are filled with waste rock from the hanging wall, sorted from the ore, from development work, or from the level above. As a rule, little timber is recovered. The examples include the Butte district, Mother Lode of California, Bunker Hill & Sullivan, Hecla, and Morning mines.

Mining Methods in the Butte District

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(New York Meeting, February, 1923)

Butte mining district is situated in Silver Bow County, in the southwestern part of Montana.

Butte originated as a placer mining camp, gold having been discovered in 1864, on Silver Bow Creek, near the southerly end of what is now Main Street. The Butte placer gold was heavily alloyed with silver, and only commanded $13 an ounce.

Placer mining was vigorously prosecuted along Silver Bow Creek and its tributary gulches for about 5 years, at the end of which time, in 1870, the Butte camp was practically deserted. In the meantime, the quartz veins prominently traceable over the hillsides were located, the first location, July 12, 1864, being made by the Missoula Mining Co. and called the Missoula Lode Claim, and later relocated by G. O. Humphreys.

Reasoning from the presence of the gold placers, free milling gold and silver ores were eagerly looked for by the early prospectors. Claims covering the great Rainbow and other silver lodes were located in the 60's. The presence of valuable silver ores in these veins caused the staking of many claims, believed to be valuable only for silver, but which later, at depth, proved to be enormous copper producers.

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[2] Mgr. of Mines, A. C. M. Co.
[3] Mgr., Davis-Daly Copper Co.
[4] Genl. Supt. of Mines, A. C. M. Co.
[5] Mgr., North Butte Mining Co.
[6] Mgr., Butte & Superior Mining Co.
[7] Asst. Mgr., East Butte Copper Mining Co.
[8] Mgr., Tuolumne Copper Mining Co.
[9] Supt. of Mines, W. A. Clark, Jr., Interests.
[10] Chief Geol., A. C. M. Co.
[11] Asst. Chief Geol., A. C. M. Co.
[12] Chief Elec. Engr., A. C. M. Co.
[13] Supt. of Machinery, A. C. M. Co.
[14] Ventilation Engr., A. C. M. Co.
[15] Safety Engr., A. C. M. Co.

Treatment of these early discovered rich ores was first attempted in arrastres. Later stamp mills were built. The first stamp mill was constructed in 1875, and others were built from time to time until a total of 290 stamps were working on silver ores. The silver-mining industry reached its zenith in 1887, and declined rapidly in the early 90's, following the sudden drop in the price of silver in 1892.

During this period of active silver mining, the business of copper mining was making rapid strides. Development of many of the veins, located as silver veins, had uncovered copper ores.

There was great activity in staking claims in the years from 1871 to 1879, when most of the claims covering the present productive copper area were located. The "outcrops", or those portions of the veins nearest the surface, were oxidized or leached, and almost without exception were barren of copper and were sought, therefore, for the silver they contained.

The Original, Parrot and Mt. Chief, located in 1864, were the first claims staked on what later proved to be valuable copper veins. The Anaconda claim was not located until 1875, when the location was made for silver ore; for a few years a large tonnage of the oxidized vein, being barren of copper, was treated in one of the silver stamp mills.

The early day copper ore mined from the Original, Parrot, Colusa, Gambetta, and Mt. Chief claims was hauled to the Union Pacific railroad at Corrine, Utah, a distance of 400 miles, whence it was shipped to eastern plants for reduction.

The first successful copper smelter in Butte was the Colorado Smelting Co. works, which treated custom ores and first operated in 1879. Following the advent of this smelter, the development of the copper mines was rapid. The discovery of rich copper ores in the Anaconda and other claims led to the formation of a number of large operating companies and soon four smelters, viz., the Parrot, Montana Copper, Clark's Colusa, and Bell, were built in rapid succession. The discovery of new producing mines quickly brought into the field the large smelting plants of the Butte & Boston, Butte Reduction Works, Montana Ore Purchasing Co., and the removal and enlargement of the Montana Copper Co.'s plant to Great Falls in 1892, under the name of the Boston & Montana Co.

In 1883, the Anaconda Mining Co. began the construction of the Anaconda smelter near Anaconda, completed in 1884, and superseded by the present works, in 1902, which is the largest copper smelter in the world. The only smelter operating in the vicinity of Butte at the present time is the Pittsmont, the property of the East Butte Copper Mining Co.

The existence of large bodies of zinc ore in some of the veins of Butte had been known for years, but until 1911 no systematic mine development for the production of zinc ores was undertaken nor the construction of any suitable milling plants. From 1911 to the present time, the production of zinc has been increasing, until this district is one of the

large zinc producers of the world. The Butte & Superior Mining Co. and the Timber Butte Milling Co. each operate complete concentrating mills for copper and zinc ores. The copper concentrates are treated at the Anaconda smelter, and the zinc concentrates at the electrolytic zinc plant at Great Falls.

Size of Mining Claims and Ownership

Prior to the Act of Congress of May 10, 1872, the rules of the district permitted the location of a claim 100 ft. wide, by 2200 ft. in length along the strike of the vein. Since May 10, 1872, claims have been 600 ft. by 1500 ft. The ownership is held in fee, by patent from the U. S. Government.

Geology of District

Granite forms the principal country rock of the Butte veins. Aplite, in the form of dikes, sills, and irregular masses, is common in the extreme western part of the district, but rarely important as a wall rock of copper veins. A number of quartz-porphyry dikes, roughly parallel to the oldest veins, occur in the central part of copper-producing area. Post-mineral rhyolite, forming plugs and dikes, occurs in the western part of the district, and in at least two instances rhyolite dikes have been observed in the deeper levels of the copper mines.

The Butte mining district covers an area 5 miles square; the copper-producing area, 2 by 3 miles, occupies the southern and central part of this area. The principal zinc mines border the copper area on the north, west and south, there being no zinc mines in the eastern part of the district. Mines operated strictly for silver ores are mainly on the north and west. Geologically, there is an apparent central copper area of zinc-free copper ores, surrounded by a zone of copper ores carrying zinc. The zinc increases in amount relative to copper outwardly from the central area, while the copper becomes less important, until it finally disappears, leaving ores of zinc with much quartz and rhodonite, but the zinc minerals are too sparsely disseminated to form commercial zinc ores, except in several important instances.

The principal copper minerals of the ores are chalcocite, bornite and enargite, with less important amounts of chalcopyrite, tennantite and covellite. In the central copper area, the gangue minerals are quartz and pyrite. In the intermediate zone, the mineralogy is similar to that of the central area, except for the additions of sphalerite.

The zinc occurs almost entirely as sphalerite and, where minable, is found associated with a gangue of quartz, iron pyrite and the manganese

minerals rhodonite or rhodochrosite, the iron pyrite, however, being much less abundant than in the copper veins. Manganese is widely distributed in the zinc veins bordering the central copper area, and in one mine is known to occur in such quantity and purity as to form an important ore of manganese. Silver is universally present in all veins.

The Butte ores occur in fissure veins of systems of great complexity. The ore-bearing veins have been divided roughly into three principal systems, known as the east-west or Anaconda, Blue, and Steward systems. These veins have been subjected to later faulting movements, resulting in many instances of a complicated arrangement of orebodies.

The Anaconda vein system comprises a great series of veins distributed over the entire district. These veins vary greatly in strike, changing from a slightly north of east direction on the west to an east-west to south-east strike in the central and eastern part of the district, respectively. The dips are generally to the south at steep angles, but many dip north, particularly in the northern part of the district. In detail, this vein system is greatly complicated through the development of important splits or branches having a northwest-southeast strike. Mineralization of a complex of east-west fissuring and associated cross fissures and branches has resulted in the great orebodies of the Leonard-West Colusa mines.

The Blue vein system includes an extensive series of veins having a northwest-southeast strike and southwest dip, later in age generally than the oldest veins of the district, but closely related genetically to the Anaconda system of fissures. This system is strongly developed in the central part of the district where it has been extremely important as a producer of copper ores.

The Steward veins are of minor importance as ore producers. They have a general northeast strike with a south dip. They appear to be later in age than the Anaconda and Blue veins.

The vein systems just described are cut by later fault fissures, one set having a northeast-southwest strike with a southeast dip, and another series, represented principally by the Rarus fault, having a northeast-southwest strike and dipping to the northwest. The displacements exhibited by these faults vary from a few feet to an extreme of approximately 300 feet.

In detail, the orebodies exhibit wide variations in structure and character, a matter of much importance from a mining point of view. Many of the large veins are continuously mineralized and minable over hundreds or even thousands of feet on strike and dip. Many of the orebodies occur in shoots, with much barren ground between. In the earliest developed veins, the valuable copper ore began at the bottom of the oxidized zone, which varies from 100 to 400 ft. in depth. Deeper developments have shown that many valuable orebodies, both copper and

zinc, have their upper limits at depths ranging from 500 to 2000 ft. from the surface.

A Butte vein of copper or zinc, in cross-section, may exhibit a single ore band, or two or more ore bands with altered granite between. The granite included between vein streaks and that forming the vein walls, usually, but not always, carries valuable mineral and frequently forms a large part by volume of the ore as shipped. Clay gouge or clay partings are common, varying from knife-blade thickness to several inches in the case of strong fault veins. Stoping widths vary from 4 to 100 ft., but most stopes are from 10 to 30 ft. wide. In the eastern part of the district there is a variation from the usual type of Butte fissure vein. In the Leonard-Tramway-West Colusa area many large orebodies are of the "stock work" type, consisting of closely spaced mineralized crisscross fissures having a northwest-southeast direction, with important mineralization of the intervening altered granite. Orebodies of this character, known as "horsetail" orebodies, are of great commercial importance.

The granite country rock is intensely altered within and along the veins, and over large areas in certain parts of the district. The alteration of the wall rocks, together with fracturing connected with the vein fissure and subsequent faulting, results in a weakened condition of the vein and wall rock, an important fact when taken in connection with mining methods.

Description of District

The Butte district covers a range of low smooth hills, sloping south and east, at an altitude of 5800 ft. above sea level. The veins dip at a steep angle, and vertical shafts with crosscuts to the veins is the system universally adopted for the development and extraction of the ores.

The climate is generally pleasant and invigorating. The air is dry and the summer is warm in daytime with cool nights. The winters are not extremely severe, and the snowfall is never such as to seriously interfere with mining operations. The Butte Water Co.'s plant furnishes an ample supply of good water for domestic and boiler uses. The mills obtain their water from small streams in the vicinity. Electric power is largely used; it is supplied by the Montana Power Co., which generates current at a number of hydroelectric plants on the Missouri River and its tributaries.

A good grade of semibituminous coal is available from Montana coal mines 250 miles from Butte. Higher grade smelting coal is obtainable from the Rock Springs field, Wyoming, 450 miles distant. Coke is supplied from Utah, Colorado, and the Crow's Nest field in British Columbia. The mining timbers used in the rough are supplied from the forests situated from 10 to 100 miles west of Butte. Lumber for shaft timbers

and construction purposes is secured from the lumber mills in the western part of the state.

The E. I. Dupont de Nemours Co. has a plant 10 miles west of Butte which manufactures the different grades of dynamite used. Construction, drilling and track steel, nails, spikes and other hardware have to be secured from the steel centers in the east.

Butte has always been able to command sufficient good labor up to the present year, but at this time there is a shortage of miners equivalent to about 20 per cent. It has its full proportion of metal miners in comparison with many other mining districts, but there is a decided shortage of miners, or men who want to follow mining, and no relief is in sight unless the restricted immigration laws are changed so as to allow men to come in from the other countries. The labor employed in and about the mines is about 35 per cent. native and 65 per cent. foreigners. The craftsmen are unionized. The miners have not been unionized for the past 8 years.

The transportation facilities are extraordinarily good. Two main transcontinental railroads pass through the district and two others have branch connections; in all, there are four of the largest railroad systems serving this locality.

The Butte district is not wet. The total amount of water pumped from all its producing properties, embracing an area of 3 by 2 miles, and with several openings 3600 ft. deep, does not exceed 5000 gal. per min., and very little increase is noted with depth. Because of the acidity of the water the pumps must be made of acid-resisting metals, and the discharge-water columns from the pumps must be lined with wood or lead.

From the copper-sulfate water pumped from the mines, over 6,000,000 lb. of copper is recovered annually, by precipitating the copper on scrap iron.

Exploration, Sampling, and Estimating Methods

Exploration

The practice of sampling and estimating preliminary to mine development in the Butte district is closely related to the geology. The deposits are veins containing oreshoots of greater or less magnitude and occurring at various depths. The veins vary in width from a few inches to 100 ft. Steep dips predominate, although dips at all angles are found. There is practically no ore at the surface except the oxidized remnants, which may contain some silver. The evidence of an oreshoot is, therefore, not the metallic contents of an outcrop, as shown by sampling, but rather its

width and structural strength, the amount and character of its oxidization, and other geologic characteristics. The veins that present the most prominent outcrops usually do so on account of a predominance of quartz or other resistant material, whereas an outcrop that originally contained more metallic filling has probably oxidized and weathered away, leaving only iron-stained patches and indistinct outcrops. Careful geologic mapping, however, serves to locate these places and they are regarded as likely areas for further development.

Churn or diamond drilling has not been found satisfactory for exploration. The evidence secured by these methods is never conclusive, even within the limits of drilling practice. Diamond-drill holes may fail to intersect an oreshoot but pass through barren points at either end of the vein. Again the drill hole may follow along the strike or down the dip of a very narrow vein, giving evidence of great width, and thus be misleading. These and similar cases have happened, so the practice of drilling prospect holes is now abandoned.

Test pitting and trenching are little used for determining mineral contents, owing to the character of the outcrops. Where the practice is resorted to, it is usually for the purpose of exposing a concealed outcrop so that it may be studied geologically, rather than for determining its metallic contents.

Tunneling is seldom done and where attempted, it is usually not to determine metallic contents but rather to seek further geologic evidence than the surface studies could disclose.

The principal methods of exploration, therefore, are shaft sinking, crosscutting, and drifting, of which the details are given later. The position of the shaft will be determined from the geologic study of the outcrop; the known extension of orebodies from adjoining underground workings or the general ore-bearing character of a certain section already determined from other underground studies.

Sampling

Little sampling is done in connection with the original determination of areas for exploration work; the methods herein described, therefore, refer to underground mining and should be considered with the section on mining.

The deposits to be sampled present a wide variety of character. Some exposures present a hard mineralization of quartz and sulfides uniformally distributed from wall to wall, but many contain bands of altered and crushed granite carrying only scattered particles of metallic sulfide. Again, a large portion of the orebodies consists of veinlets of chalcocite and enargite traversing granite in such close succession as to make the whole mass valuable as ore. The moisture content does not

affect the methods of sampling. It amounts to 2.5 per cent. on the average of mine run product at the smelter.

The method of taking and averaging samples is about the same throughout the district, except in the case of one company. The details for the principal producing companies follow:

With one company, all sampling is done under the direction of the geological department. The sampling branch consists of one or two samplers at each mine, and a chief sampler, whose headquarters are in the geological office. His instructions to the samplers are issued with the approval of the chief or assistant chief geologist. This department has classified the exposure into a few standard types and has shown them on simple drawings, which are given to the samplers for their guidance. In this way the sampling is standardized in all the mines, and the personal equation reduced to a minimum. The samplers make daily visits to all drifts and raises and also take any samples in the stopes that the mine foreman may require. The ideal cut or channel across the face of the drift is impractical from an operating point of view. Too much time would be lost if the sampler laid out and chiseled a neat groove across the vein. The same principles apply, but the result is accomplished by the use of the sampler's pick. The sample is caught in the sack, which is held open by a wire ring with a handle fastened to it. The sacks are 6 by 14 in. The samples taken any day are sent to the assay laboratory and results returned to the sampler and mine office next day. He keeps a record of results in a book prepared for the purpose, and submits a daily report to the geological department, where it is used to keep in close touch with the development work. The weekly report ends with the work done each Wednesday, so that the assays are available and the averages figured for delivery on Friday.

The method of figuring averages has been carefully standardized and type forms have been given to the samplers to show different cases. All averages are determined by the usual method of multiplying widths by assays. The results of the daily reports are copied into loose-leaf ledgers. The averages for the week are plotted on cards ruled to tenths of an inch. Each drift and raise has a card, properly identified by number and name of the mine. The horizontal scale of the cards is 1 in. equal to 50 ft., which is the same as the standard geologic maps. The vertical scale is 1 in. equal 10 per cent. for copper or 10 oz. for silver, and the average widths are marked at the proper points. From these curves, the averages for any length or part of any drift or raise are easily estimated, and by placing the assay card for any drift under the tracing of the corresponding geologic map, the latter becomes an assay map. These records are used in the calculation of ore reserves, as described below.

Another company employs two samplers at its mine, who are under the direction of the engineering department, which also does the geologi-

cal work. It takes each sampler four days to visit and sample all the working places. One starts two days after the other, but each covers the entire mine. The first sampler posts a card showing width of sample at the place from which he took it. By the time the second sampler reaches this point, the results are available and he writes in the assay on the card. This procedure applies particularly to stopes. One day a week is called development day, when the samplers visit all the drifts and raises. The samples are taken with a sampler's pick and caught in a canvas held by a light frame, from which it is poured into a sample sack about 5 by 12 in. The records consist of a loose-leaf ledger, in which all drift and development samples are posted. The stope samples are posted on a set of 100-ft. longitudinal sections by means of a simple set of conventional patterns, which give both width and assay. These records are used in estimating ore reserves.

Another company employs two samplers, who visit all the working places as often as they can. They are responsible to the mining department and take orders from the superintendent. They receive some instruction from the geologist and send him a copy of their reports. The samples are taken with a sampler's pick and collected in a canvas catcher, which is supported by a light frame. The sample sacks are 11 by 14 in. The records consist of assay maps separate from the geologic or engineering sets. These are surveyed maps for the levels, but are diagrammatic maps on cross-section paper for the stopes. The results of drift assays are also posted in a ledger. The results of stope assays are copied into small books for the respective shift bosses.

Another company employs two samplers. They are under the engineering department, but send a copy of their reports to the geologist, who makes suggestions to them. The samplers make daily visits to all working places, both stopes and development work. The samples are caught in a canvas device supported by a collapsible frame and transferred to sacks 7 by 14 in. The sampler puts up a numbered tag at the location of the sample in the mine and the assay results are copied into books under corresponding numbers. Further records consist of assay plan maps of the levels, upon which the assay results are written. The stope samples are shown in figures on 20-ft. scale longitudinal sections.

Another company's sampling methods differ materially from those of other properties in the district. No samplers are employed and no regular samples are taken from working places. Each working place has a number. When the ore from any place reaches the shaft, the station tender notes the number of the place from which it came; when he rings the hoisting signal, he also signals this number to the topman so that it will be known on top from what place each skip load of ore came. As the skip discharges into the surface tram car, a sample is cut out by a divider placed over the car. This sample is then placed in a box, with a number

corresponding with number of the place from which the ore came, and at the end of the day all samples so collected are sent to the sampling mill, to be cut down and prepared for assaying. The results are plotted on longitudinal sections and listed in books.

Another company follows the general methods of the district. The samplers make daily trips and take samples with a hand pick. The results are plotted on longitudinal sections by the use of a legend, which shows the product of width by assay, rather than width and assay separately.

Estimating

The methods of estimating tonnage of orebodies are similar throughout the district, with the exception of one company. Details for the principal producing companies follow:

With one company the geological department has charge of the estimating of ore reserves. For this purpose a longitudinal section of each vein is made on a scale of 1 in. equals 100 ft. and projected on a vertical plane parallel to the strike of the vein. A certain minimum width and assay value are assigned by the operating department as the lower limit for ore. Then the drifts and raises on these sections are colored red wherever the assay-curve records, above described, show the required grade of copper ore, and green for zinc ore. Drifts and raises, in waste, are colored gray. This color scheme shows quite clearly what should be included in the ore reserve for each section. Blocks are then laid out and their areas determined with a planimeter. Average widths are taken from assay records or actual stope measurements when available. The best average relation of volume to tonnage is found to be 10 cu. ft. equal 1 ton. Variations from this figure are not less than 9, nor more than 11. For the grade of these blocks, properly determined averages are calculated either from the assay-curve records of the drifts and raises or from the more complete daily reports in the ledgers where necessary The outlines of the blocks are laid out with due consideration for the pitch of the oreshoot, the losses of waste it may contain and other irregularities.

The estimating of ore reserves for another company is done by the engineering department, with the assistance of the geologist. The work is done on 100-ft. scale longitudinal sections and does not differ materially from the method just described, except for the assay records described under sampling.

The method of estimating tonnages with another company differs from general practice in the district in that it is done from plan maps upon which the assay results have been plotted, instead of longitudinal sections. The horizontal areas of ore are multiplied by the vertical distances between them to get the volume.

Another company estimates its ore tonnages by means of 100-ft. scale longitudinal sections prepared by the engineering department, each section covering one vein. Otherwise the method differs only in slight detail from the usual Butte practice.

Two other companies use the longitudinal section method of estimating ore reserves. The scale is 1 in. equal 100 ft. The work is done by the engineering department, with the assistance of the geologists.

There is, naturally, some difference between the estimated grade of ore blocks and the grade of ore actually mined, but these differences depend largely on the width and character of the veins. These characteristics are known from the geologic notes. In the case of uniform ore exposed in several drifts where the widths were suitable for convenient mining and the walls definite, the methods outlined are very accurate. It was found by checking the grade of ore shipped from several drifts of this kind against the estimated average for the same drifts that an error of only 0.5 per cent. was found in both copper and silver. In most cases, however, the ore is not uniform in character and the assay widths are not suitable for mining. In these cases, the assay widths are increased to fit the mining practice and the corresponding decrease figured in the grade. By this means, the estimates are believed to be close to the grade of the ore shipped, but no checks are made against the ore shipped except in a few cases, above cited.

The accuracy of tonnage estimates varies considerably, although, in many cases, the figures have been close to the final result of mining. The estimates are made once a year and, in cases where unforeseen conditions have arisen, new blocks are outlined so as to keep the error of the total as low as possible.

History of Principal Mining Methods Formerly in Use

Mine openings in the Butte district consist almost entirely of vertical shafts, the deepest of which is 3633 ft. from collar to sump. There are a few incline shafts, which are used for ventilation only.

Shafts are sunk, if possible, on the foot-wall side of veins and in blocks of ground not traversed by veins or fault fissures.

In the larger mines, the shaft usually consists of three and one-half compartments, the two main compartments being used for the handling of rock, while the third compartment is used for transferring men, tools and supplies from the surface to the various levels without interrupting the main hoisting operations. Air and water columns, electric power and signal cable, and ladderway are carried in the half compartment. A complete description of shaft-sinking methods of Butte will be found in the *Transactions*,[1] and under the heading "Underground Mines."

[1] *Trans.* (1913) **46**, 151.

Stations are cut at regular intervals of from 100 to 200 ft. From the stations, crosscuts are driven to intersect the veins from which drifts are extended along the strike of the veins.

The method of opening up the sills, as well as the method of mining followed thereafter, varies considerably in veins of different characters and different widths. The method formerly followed in working out sills on large veins consisted of removing all of the ore on the sill between the foot and hanging walls, square setting and filling, leaving a drift one set wide open on the foot-wall side. In particularly heavy ground, it later became the practice to open the sill with a drift but one set wide, leaving the remainder of the ore on the sill to be extracted from the level below. As soon as a sill is opened, raises are driven to the level above for ventilation, lowering of tools, timber and waste for filling as stoping progresses. Chutes and manways are carried up on the foot wall at intervals of from five to eight sets. In the larger orebodies, these raises usually consist of two chutes with a manway between.

As stoping progresses, it is often difficult to maintain drifts within the walls of the vein, even though the drift is bricked and sheeted. Before it becomes necessary to abandon a drift that is taking excessive weight, a lateral is driven from the main crosscut in the foot wall of the vein and parallel thereto. The distance of the lateral from the vein is usually from 20 to 40 ft.; and from the lateral, crosscuts are driven to the raises and chutes extending from the sill to the stopes above. In some instances, raises are driven from the lateral to connect with the original raises driven from the sills, connection being made usually with the original raises at the fourth or fifth floor.

Several methods of stoping are followed, but all stopes require refilling with waste to prevent caving. Overhand stoping is the general method followed throughout this district. The method originally used was the horizontal cut with square-set timbering carried to the back and face and filling following within two floors of the back. Later, where conditions permitted, the horizontal cut and back-filling system was employed without timber. A description of this method may be found in *Transactions*.[2]

Following the "back filling system," rill stoping was later adopted. Inclined cut-and-fill, or rill stoping, with and without timber, is the method now being employed in many of the mines. This is merely a modification of the original methods, except that the back is cut and the filling below is carried on an incline sufficiently steep to permit the broken ore to run to the chute by gravity. After all broken ore has been drawn from a stope, it is filled with waste from a chute extending to the level above, after which another cut is taken from the back.

[2] B. H. Dunshee: Timbering in the Butte Mines. *Trans.* (1913) **46**, 146.

In mining veins 15 ft. or less in width, the sill is always worked out for its entire width and may be timbered with square sets or heavy drift sets with caps of various lengths extending from foot to hanging wall. Where drift sets are stood, round timber is used, varying in diameter from 12 to 18 in., and square sets are stood on the first floor. Various forms of sheeting are used to take the weight off the center of sill caps and heavy lagging laid to carry the waste with which the stopes are filled.

Waste for filling is secured from development work, sorting in the stopes and waste drifts driven off the floors into the wall or country rock. In some instances, waste is procured from old mine dumps and lowered either through chutes from the surface, or through the shaft, to various levels where it is distributed to the various stopes. In areas containing mine fires, concentrator tailings are being effectively used in filling old gobs and smothering fires that could not otherwise be reached.

The methods described are applied generally in mining the copper and zinc orebodies, and brief mention may be made of the methods employed in working the small silver veins. The higher grade silver ores developed in the northern and western portion of the district usually occur in veins of narrow width, contain much quartz, and can often be stoped without timber except for an occasional stull from foot to hanging wall. In veins of this character, the ore is often stripped where narrow, the waste being used for filling. Where the orebodies are over a set wide and require timbering, or where the walls must be supported to prevent caving, methods similar to those employed in the copper mines are adopted.

Important Features that Affect the Operation of Mining Methods

Although in some mines the ores are valuable principally for their copper content, in others for their zinc content, and in others for their silver content, most of the important variations of the conditions and methods of mining occur in the mining of the copper ores, so that a description of these will include all operations.

Practically all of the important deposits occur in granite or aplite, in fissure veins in which the sulfide minerals and quartz and, in some cases, rhodonite and rhodochrosite and unimportant amounts of other gangue minerals have largely replaced the original rock. In many parts of the district, the country rock, for considerable distances from the walls of the vein, has been considerably altered and decomposed by hydrothermal activity. This is especially true along the joint planes, in which this alteration, together with the adjusting movement that has taken place, have so weakened the structure that the walls of the stopes, or even of the drifts or crosscuts, are apt to lack self-supporting cohesiveness so that they

crumble in blocky masses. This tendency is greatly magnified by even small quantities of water percolating through the country rock or finding a way through the stopes from levels above. The continuity of rock support is practically destroyed in the planes of the numerous and complex post mineral faults that are manifested in most cases by impalpable gouge seams. Frequently these follow the walls of the veins, but in many cases leave them at acute angles to cut either into the vein or the country, often to return again to the wall of the vein. These conditions cause much sloughing and squeezing of the walls and back and demand close timbering, filling of stopes before long exposure to the air, frequent cleaning away of broken-down material in stopes, crosscuts and drifts, much repair and replacement of timbering and construction of bulkheads. In order to be supported, drifts and crosscuts must be driven of small size, which, in turn, prevents the use of large cars, locomotives, or mechanical loaders. These conditions also prevent, except in unusual cases in a few small veins, the adoption of stull timbering, open stoping, shrinkage stoping or caving methods, and restrict the size of shaft compartments and loading pockets that may be adopted without danger of prohibitive expense for jacketsetting and other maintenance. For the same reasons and on account of the limited life of timber, it is impracticable to explore or develop ore reserves extensively, so that plans for mining, ventilation, etc., cannot usually be made with any reliable knowledge of the size or shape of the orebodies.

The amount of exploration or waste development that can be done to advantage is governed by the ability to dispose of the waste without hoisting to the surface, by use for filling in stopes. This, in turn, is in inverse proportion to the amount of waste that can be, or is, left in the stopes as a result of separation from the broken ore. As a general rule, about 25 per cent. of the ore that is hoisted comes from drifting, silling, or raising, in ore and to keep stopes filled and to maintain ore reserves it usually requires about 1 lin. ft. of development advance in ore and waste for each 22 tons of ore hoisted. From 25 to 50 per cent. of the development advance is in ore and the balance in waste, and, of every 20 tons of ore hoisted to the surface, there are on the average about 15 tons from the stopes and 5 tons from development and there are left broken in the mine about 3 tons of waste from development and about 4 tons of waste sorted from stope ore. These proportions vary considerably, depending on the character of the vein.

The vein systems are complex, showing much variation in strike and dip and displacements in many places of the veins and orebodies for distances up to 300 ft. by post mineral faults cutting the veins at all varieties of angles, often dividing the orebodies into separated blocks of ore of relatively small dimensions, thus greatly complicating the development and the location of chutes and manways, stoping, timbering, etc.

Vein dips, however, are seldom flatter than 40° from the horizontal, so that the ore, unless wet, will in nearly all cases run by gravity on the slope of the foot wall of the vein. In most cases the walls of the vein are fairly distinct, though occasional irregular mineralization extends into the wall rock. Except where the vein is broken by faulting, the width of the oreshoot is usually small in proportion to the length and depth. Many of the orebodies, however, exceed 12 to 15 ft. in width and up to widths as great as 100 ft. so that single timbers are insufficient in length to make suitable caps or stulls for timbering in a large proportion of the stoping, this resulting in the general adoption of square-set methods of mining, except for the narrow veins that have walls suitable for rill stoping. Oreshoots show no pronounced tendency for the depth greatly to exceed the length, or vice versa. The ore is likely to be considerably wider and of better grade at the intersections of veins, either on dip or strike. In the larger vein systems, the commercial ore frequently occurs in shoots occupying only a portion of the width of the vein, the remainder being too low grade to be commercial. Frequently, roughly parallel shoots occur in the vein, one of which may be against the foot wall and the other against the hanging wall, with sufficient mineralization between the two in some places to make continuous ore. In other instances, the two walls of the vein may be connected by commercial oreshoots transverse on either dip or strike, or both. The ore in many of the veins is apt to occur in lenticular form, the vein in the barren portions frequently pinching out to such small proportions that it would not be recognized as being of importance. The largest oreshoot of the district is in the Original vein system of the Anaconda Co.; it varies in width from about 10 to 100 ft., probably averaging 25 ft., and with a continuously stoped length of 7700 ft., and a developed depth of about 3600 ft., with the bottom as yet undeveloped. The complication of conditions and the resultant uncertainty of shape and location of branching or faulted portions of oreshoots, makes it impracticable in most cases to follow, without deviation, any orderly plan of development or mining, and frequent intermediate levels and developments are necessary.

The mineral contents of the hoisted ores of the district are quite variable and depend to some extent on the subsequent treatment. The copper ores of two companies are all concentrated before smelting, so that no segregation is now made into first and second class. Part of the copper ores of two other companies are smelted direct in the blast furnace, and at these places closer sorting is practiced, either in the stopes or after hoisting to the surface, for the purpose of segregating the direct smelting ore from the milling ore. At the same time the waste is carefully eliminated. At two other company properties, all the zinc ores are concentrated so that no segregation is attempted. At two small

mines of two companies, the ore production varies from thoroughly oxidized silver ores to silver, zinc, lead sulfides ores, and the oxidized ores are segregated from the sulfide ores, the oxidized ores being smelted direct for their silver content and the sulfide ores concentrated. No segregation into classes is made of the sulfide ore.

In most of the above ores, the sulfide mineral content varies from 25 to 35 per cent. of the weight, the remainder being gangue, principally quartz, with some of the granite or aplite feldspar. In very few cases is there any considerable enrichment of the valuable minerals into the fines, so that no especial precautions are taken for the prevention of loss of other than reasonably close-laid flooring in the stopes. In the instances where it is important to segregate ore into two classes and to make a good separation from the waste or low-grade ore, the difficulty of handling becomes greater, as chutes must be provided for both classes, and storage becomes harder to arrange.

The methods and costs of mining are affected in the copper mines of the district by the warm copper-acid water encountered. Most pump and drain lines and pump parts must be constructed of bronze, or lead or wood lined. Air and water lines and tracking, pipe fittings, rock-drill parts, etc., require, in many instances, special preparation by painting, wrapping with tarred canvas, etc., and, in spite of such precautions, may require frequent replacements. Hoisting ropes, cages, skips, mine cars, etc., must be built especially strong and satisfactorily heavy, and frequently painted and repaired, to withstand the corrosive action of the water. Even the moisture entrained in the ventilating air is especially corrosive, this being indicated by the rapid corrosion of fans, hoisting cables, and other parts exposed to its influence.

In the lower levels of the Butte mines, the rock and water temperatures are high, reaching on the average about 100° F. at the elevation of the 3000-ft. level of one company's workings, with an increase of about 1° for about 100 ft. beneath this level. This high rock temperature, together with the high temperatures resulting from the decomposition of rock filling and timbers in the old workings of the higher levels, demands extensive and careful ventilation engineering, and requires the direction of considerable volumes of the best available air to each and every working face, whether on the sill or in the stopes. There are no great differences of elevation of shaft collars, so that most of the circulation of the air must be done by mechanical means; it is the practice at nearly every principal operating mine to install a reversible fan of large volume over a shaft used exclusively for ventilation purposes. These fans are, in most cases, operated as exhaust fans so that the hoisting and operating shaft may have the benefit of the downcast current of cool air. Many smaller auxiliary fans are utilized underground to divert the air from the main courses to the headings at which it is required, usually by forcing it through

specially prepared fabric tubing of variable sizes. The high temperatures underground cause greater evaporation of the water, resulting in higher humidity, which demands a larger volume of air. This has a tendency, in turn, to create dust through the workings, which again necessitates sprinkling to lay the dust. The increased humidity of the air also results in faster decay of the timbering, necessitating protective measures, such as guniting, timber treatment, etc.

On account of the complexity of the geological structure, unusually comprehensive surveys and geological mapping must be made, and on account of the large amount of valueless sulfides in the copper ores, it is difficult to determine visually the relative values of the ores, so that careful and systematic sampling and assaying are necessary.

Mine Openings, Shafts, or Tunnels

Mining operations are carried on through vertical shafts. This type of shaft was adopted instead of the incline shaft for the following reasons: The steep angle of dip of the veins; the heavy ground in and near the veins, so that repair costs would be much higher in inclines; several decks of cages can be used in lowering timber and handling men; and cages can be easily and quickly changed to skips when ore is to be hoisted.

Size of Shafts.—The size of shafts varies somewhat, but the standard shaft is rectangular in shape, 20 ft. long by 7 ft. wide, outside dimensions. It is divided into two hoisting compartments, an auxiliary compartment, and a pump compartment. The two hoisting compartments and the auxiliary compartment are each 5 by 4½ ft. and the pump compartment is 5 by 3 ft., inside measurement. Sketches of shaft timber are given in the *Transactions.*[3]

Hoisting and lowering is done in counterbalance in the two hoisting compartments, and the auxiliary compartment is used for lowering timber and other supplies, and for handling men coming on or going off shift or during the shift. In the counterbalance compartments, skips and cages are interchangeable, so that cages can be used for lowering and hoisting men going on or off shift and can also be used part of the shift for handling supplies. The fourth compartment is used for water columns, air lines, electric cables and manway. Loading and discharging skip pockets are provided on all hoisting levels.

Depth.—The depths of the shafts vary from 800 to 3600 ft.

Construction.—The shafts are all constructed of timber, gunited timber, or timber and concrete slabs. One shaft is partly constructed of steel I beams and reinforced concrete.

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[3] Norman B. Braly: Shaft Sinking Methods in Butte. *Trans.* (1913) **46**, 151.

Reasons for Location.—In selecting a permanent shaft location, the principle facts taken into consideration are: Rock formation free of faults and veins, whenever possible, and in ground showing minimum movement; close proximity to large orebodies; due consideration of the dip of the veins and faults; location suitable for construction of permanent engine, head frame and ore-bin foundations; site giving waste dump room and the probable effect of underground operations with respect to the movement of the ground at the surface.

Each mine is provided with one or more ventilation shafts through which the air is drawn out of the mine. Most of these shafts are smooth-surfaced and fireproofed with concrete slabs.

Underground Development Plans

The general plan of development consists essentially in opening up levels at 100- or 200-ft. intervals (dependent on whether the veins are wide or narrow) and crosscutting the shortest distance from the shaft to the projected position of the vein or orebody it is desired to open, and then to turn and open the vein both ways by means of a drift and short crosscuts from the drift to the walls of the vein. The position of the drift with relation to the walls of the vein is dependent on the character of the walls, the width of the vein, and the dip of the vein. After the level development is complete or simultaneously with it, raises are driven through from level to level, primarily for ventilation, and to block out the ore, but later to serve as waste chutes for filling the stope below, handling ore and for lowering supplies into the stope. After the raises are through, chutes are built along the drifts or crosscuts in the vein at 25- to 40-ft. intervals, and stoping started. Twenty-two tons of ore are developed by 1 ft. of development work.

The general system of mining is square set, fully 80 per cent. of the ore being extracted by this method. The second most important system of mining is the rill stope.

The use of sublevels or intermediates in the Butte district is confined to special cases, such as making connections to other mines when the elevation of the levels are different, development of fault blocks of ore, development of the bottom of an oreshoot that does not reach down to the main level, and the development of remote bodies of ore that would require a long crosscut from the shaft to open them up according to the general plan used on main levels. As far as possible, their use is avoided, for it means rehandling of the ore and more labor in getting supplies into the working places, which means a higher cost per ton of ore extracted.

The plans of development, when considered in detail, classify themselves into two groups: Development on wide veins (20 ft., or 4 sets, and

wider), and development on narrow veins (from 12 in. to 20 ft. in width). This grouping will naturally overlap in many cases.

Development Plan for Wide Veins.—On account of the size of the orebodies and width of the vein and the heavy character of the ground when being mined, it is necessary that the orebody be divided into sections and that each section of the ore be mined as rapidly as possible, in order to keep repair costs as low as possible and to prevent caving of the ground. The general plan is to use low lifts and to divide the orebody into blocks, much the same as a checkerboard, and then to mine alternate blocks of ore, fill these mined-out sections with waste, allow the waste to settle, and then to mine out the blocks that have been left. In mining these blocks, the aim is to put the chutes or raises as close together as possible so that all the ore may be slid into the chutes by slides and the waste filling distributed all over the stope without shoveling. These blocks are mined by the square-set system.

The general plan of development is as follows: Levels are opened up from the shaft at 100-ft. intervals, and a crosscut driven the shortest distance to the orebody. After the crosscut reaches the vein, the main drift is driven in the foot wall, if the vein is very wide and has heavy walls, or driven on the foot wall if the vein is 30 to 50 ft. in width and has fairly good walls for supporting the drift. From this foot-wall drift or lateral, as the case may be, crosscuts are driven to the hanging wall at intervals of 25 to 50 ft. This interval, of course, is largely governed by the specific conditions existing at that place. From these crosscuts, raises are driven to the level above. The spacing between raises varies, but is generally about 25 to 30 ft. (about 20 ft. has been found to be the minimum spacing that can be used, because with closer spacing the pillars of waste filling in the mined-out sections are so narrow that they become unstable and crush the chutes). Each chute or raise is used in mining out four blocks of ground, the blocks cornering at the raises. The above plan of development must be considered only in principle, there being many variations at the different mines according to the specific conditions encountered.

Development Plan for Narrow Veins.—In developing narrow veins, the standard interval between levels is 200 ft. This spacing between levels is considered to be the most economic interval.

On each level, the vein is opened by a crosscut from the shaft, driven directly to the projected position of the orebody. After the vein is reached, it is opened by a drift, which may be driven on the foot wall of the vein, if the vein is wider than one set, or directly on the vein, if it is the width of a set or narrower, or driven with the vein on the hanging wall, if it is a very narrow vein. Local conditions at each property determine this. However, the most general practice is to drive directly upon the vein, the veins being generally the full width of the drift. After

the drift is completed, or simultaneously with it, main raises are driven through to the level above at intervals of 100 to 150 ft. if the orebody is continuous, or else at irregular intervals if the ore is located in shoots. These raises are driven primarily for ventilation and blocking out the ore, but later are used for lowering supplies into the stopes and as waste chutes for running filling into the stopes from the level above. After these raises are through, stoping is started and chutes built at intervals of 25 to 40 ft. along the drift. These chutes are carried up with the stope as mining progresses.

Dimensions of Development Openings.[4]—Throughout the whole of the district, virtually no distinction is made in the sizes of crosscuts or drifts, on account of the type of haulage used. The dimensions of the drifts and crosscuts vary within a range of 6 in. from company to company. The tabulated statement of the principal dimensions of drifts and crosscuts here given is an approximate average. This size of drift or crosscut is used wherever sill openings are made, regardless of whether motor or mule haulage or hand tramming is used. The principal type of haulage on main levels is by motor, with the exception of two mines, which use mule haulage. The average size of car used is 14 cu. ft., being 26 in. wide, 40 in. long, 24 in. deep, and 45 in. high.

Clearance width at cap	4 ft.
Clearance width at height of a car	5 ft.
Clearance height	7½ ft.
Spacing of sets, center to center	64 in.
Batter on posts	6 to 12 in.

In driving drifts on veins where it is expected to sill out, the posts are often stood plumb and a 5-ft. 10-in. cap used, giving 5 ft. in the clear between posts.

Untimbered crosscuts are used whenever the ground is solid and stands well. In driving them, the general shape of a cross-section of the opening is square in the bottom, with a slightly rounded toe and an arched back. Some mines use an approximately square back also, so as to provide room for ventilation pipe. They are driven from 7 to 9 ft. high and 6 to 7 ft. wide.

Main raises from level to level are driven for ventilation, for handling ore, to give access to stopes for men and supplies from the level above, and to dump waste through from the level above to the stope below for filling. They are sometimes driven with a manway and chute compartment, but

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[4] Plans of development on main levels are shown in Ore Deposits at Butte, Mont., by Reno H. Sales, *Trans.* (1913) **26**, 3; and Applied Geology in the Butte Mines, by Frank A. Linforth, *loc. cit.*, 110.

in most cases, and in the case of rill stopes or when other conditions demand, they are driven three compartments wide, consisting of two chutes and a manway compartment in the middle. They are all square-set except those raises through which it is expected to pass a large quantity of rock. These last are cribbed or bricked with timber. Raises are usually offset one set into the foot wall from the drift and then driven with vertical posts offsetting to keep the foot wall as the raise progresses. An approximate average of the size of compartments in raises is:

Manway compartment inside	52 by 52 in.
Chute compartment inside of lining	48 by 48 in.
Sets from center to center of posts	64 by 64 in.

Variations of 6 in. from these dimensions may be found at the different mines.

The same statement and dimensions as made for raises can be applied to chutes, the only difference being that chutes are carried up as the stope progresses, and the raises are driven directlv through from level to level. The interval between chutes varies from 25 to 40 ft.

Trackage.—Four companies use motor haulage and two use mule haulage. All use hand tramming for very short distances. The weights of motors used vary from 3½ to 4¼ tons.

Cars vary in weight from 765 to 1030 lb., and have from 13 to 16 cu. ft. capacity, the average being 14 cu. ft. The weight of car and load will vary from 2200 to 2500 pounds.

The same gage of track is employed for all types of haulage. All companies but one use 18 in.; this company uses 20 in. One of the companies operates two of its mines with 20 in. The average grade of track in the district is 0.5 per cent. The curves are laid out on a minimum radius of 18 ft. All companies using motors use 25-lb. rail; for mule haulage, and hand tramming, all companies use 16-lb. rail.

Drainage of Levels.—Some ditches are carried in the center of the drift beneath the track ties, but the general tendency is toward using the side ditch, wherever admissible, because it is more easily cleaned and maintained. Wherever drainage water must be carried past workings opening to the levels below, it is customary to carry it past these places in wooden boxes or pipes to prevent seepage into the workings.

Wherever a large volume of water is to be carried, drainage drifts are used. These are simply a plain drift or crosscut, with a false planked floor on stulls raised above the bottom. The water flows beneath the floor.

Sizes of Compressed-air Lines.—Small mines use 6-in. pipe and large mines use 10-in. pipe; the general practice is an 8-in. column. For main feed lines on levels 3- and 4-in. pipe is used. The general practice is to use 2-in. pipe for branch lines on levels beneath stopes, but some mines use 4-in. pipe.

The sizes of branch lines from drifts up into stopes vary considerably. Two companies use 1½-in. pipe, another 1¼-in., two others 2-in., and one 1-in. pipe.

For driving drifts and crosscuts, 2-in. line is almost universally used. However, the ultimate object of the drift or crosscut often determines the size of air line laid, it being more economical to lay the permanent line eventually required, at the time the work is done. For driving raises, the various companies use the same sizes as are used in the stopes.

The systems on each level are independent of one another with the exception of connections made through the raises. The customary practice, when a raise reaches the next level, is to connect the 1½- or 2-in. pipe used in driving the raise into the main air line on the upper level.

Sizes of Ventilating Pipe.—All companies are using canvas pipe, made up in 50-ft. lengths. The sizes of pipe used vary with the length of run and the amount of air desired. However, under ordinary conditions, the following may be considered as general practice: For ventilating isolated sections of the mine, 24-in. pipe is used; however, this size is used only occasionally, because connections are driven as soon as possible to relieve this condition. For ventilating long drifts, crosscuts, and stopes, 16-in. pipe is used. For ventilating short drifts and crosscuts and the backs of raises, 8- and 12-in. pipe is used.

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Mining

Drilling and Blasting.—Standardization of rock drills, steel, or methods has not been accomplished over the Butte district, as different companies vary considerably, not only in equipment but in the methods of drilling and blasting used.

The drills used for drifting seem to be most generally the No. 248 Leyner Ingersoll, Sullivan DX-61 and the Waugh Turbro No. 21-V. In the stopes, the Ingersoll-Rand CCW-11, Sullivan D. T. 421, CCW-21 and BCW-21, and Waugh 16-V are used. Some mines use dry stopers, most of them use the hand-rotating wet stopers, and two use the self-rotating wet stopers. It is the intention to make a standard of the self-rotating wet stoper. The jackhamer type drills are usually the Waugh 55 Clippers, or the Ingersoll-Rand BCR-430 Jackhamer.

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Drill steel, likewise, has its variations: One company uses 1-in. quarter-octagon for both drifting and stoping, with the regular double-taper cross bit. Another company drifts with 1¼-in. hollow-round and stopes with hexagon or cruciform, but practice here is toward the adoption of ⅞- and 1-in. quarter-octagon for the stopers. Here, too, the double taper cross bit is standard, with the 5° and 14° angles. Another company is at present changing from 1¼-in. hollow-round to the 1-in. quarter-octagon; hence will have but one steel for both stopes and sill work. Recent tests on a modified McClellan bit of the double-well type has led to its adoption on this property. Another company standardizes on 1¼-in. hollow-round for drifting and 1¼-in. cruciform for stoping and here the double-arc bit is used. Another company has the same practice, but uses the double-taper cross bit. The practice here, however, is leading to the use of 1-in. quarter-octagon for the drifters and stopers. Another company follows the practice of 1¼-in. hollow-round for drifting, 1¼-in. cruciform and 1-in. quarter-octagon for stoping. A double-taper bit, with angles of 10° and 30°, is used here. Jackhamer steel over the whole district is the ⅞-in. hexagon.

In drilling in drifts and crosscuts, the general practice is the use of the usual downcut round of from nine to twelve or fourteen holes. This round is drilled from one set up on the bar, with the exception of the final drilling of the lifters. In the stopes, the practice is all stoper work or part stoper and part Leyner. In the former case, all holes are put in from the set below, as near vertical holes, or they are placed at angles to the vertical face of the ore. Where the Leyner is used in the stopes, both horizontal and vertical holes are used, so that this method resorts either to straight breast stoping or a combination of both breast and overhead or back stoping. In any event, the ground for a set is broken with from five to seven holes on an average.

For tamping, the general practice is to use mill tailings compressed in regular tamping bags, or clay or gouge from seams along the drift. The definite practice is to use three sticks of tamping to every hole in sill work and two sticks of tamping to each hole in the stopes. Subsequent bulldozing at grizzlies or chutes is only done as necessary by men in the stopes. A few mines find it necessary to have a man making the rounds of a stope with a jackhamer, plugging boulders, but this is not general practice.

The grade of explosives is usually 33-per cent. ammonia powder in the stopes, 40-per cent. gelatin in raises and drifts, and 60-per cent. gelatin for sinking. No. 8X blasting caps and Lion Brand and Comet Special fuse are used. In shaft sinking, electric detonators are used.

Minimizing the loss of fines does not play a part in Butte blasting practice, except in cases where an excess amount of fines is an

indicator of the use of too much powder for a round. This is automatically taken care of in the case of the standard rounds spoken of above.[5]

Drifting and Stoping.—Levels throughout the district are generally spaced 200 ft. apart, vertically, and drifts are driven on the veins. The 100-ft. interval is used on some of the properties. The headings are generally about 5 by 7 ft. in the clear and, except in cases where laterals are used in wide orebodies, the aim is to maintain the drift as the main haulageway. General practice is to break 5 to 6 ft. per round, but some operators believe the shorter, say 3½-ft. round, drilled, blasted and mucked every shift is preferable.

Drilling is done generally with the vertical bar with either two or three settings of the arm, but one property consistently uses the horizontal bar so that mucking can be carried on simultaneously with the drilling. (In this case, they are pulling the shorter round.) It is general practice to offset opening drifts for rill stopes, at intervals for raises. Three sets are left open to furnish two chute compartments and a manway. The raise or chute interval with rill stoping varies in different mines; while some adopt the general practice of each 50 ft., most rills have raise spacing near to 120 or even 150 ft. In planning for square-set stopes, the raise interval (or chute interval, as chutes are carried up with a manway) is generally from five to ten sets. The general practice is to aim at about every five sets (25 ft.). Raises and chutes are usually on the foot-wall side of the drift.

Sublevels seem to be used mainly for the prospecting of the walls of the stopes and of the faulted sections of the vein. There is no definite practice as to their use, and geological conditions or the need of filling govern their location entirely.

In untimbered rill stopes, the entire back is drilled in rows of 6-ft. holes, from foot to hanging wall, with a spacing of 2 to 3 ft. between the rows. The sequence of operations is to fill the stope within 3 or 5 ft. of the back, and then complete the drilling of the stope. The floors are then laid and the whole back blasted. After barring down, the ore is run through the grizzlies or the men complete the drilling of the stope, draw the ore, pick up floors, run in filling, lay floors, blast and bar down. Work is alternated from one side of the rill to the other, so that the backs of the two stopes are kept even. Stulls are used for heavy walls. Ore is also mined by the rill method where square-set timber is used.

Square setting is used where the shape and size of the orebodies, or the walls, are of such character that rilling is not practical. At the present time, this is the chief system of stoping in the district. In opening a square-set stope, in every third or fourth offset left for the raises, raises are started through to the level above; and when they have pro-

[5] Arrangement of holes is shown in Standardizing by North Butte Mining Co., by Robert Linton. *Trans.* (1921) **66**, 182.

gressed far enough not to interfere with the work in the stope, the first two floors are mined off through stop boards, the muck being dropped into cars along the drift. Sheeting is then put in place above the sill timbers, and the chutes with manways between the raises are started up with the stope.

Waste filling is obtained by either crosscutting the walls of a stope, incidently prospecting the walls, or it is furnished from development work on the levels. The latter case is the most common supply. The filling is dropped into the chutes from the level above, drawn off from a temporary chute near the top of the stope, and permitted to run into place, or distributed in cars to the points needed. Cross cutting the walls weakens them and hence is not desirable, except in cases of isolated or absolute need where other waste would be costly to obtain. In some cases, in the mining of wide veins, it has been necessary to use surface or underground glory holes for a supply of filling.

Sorting the ore in the stopes also furnishes some waste, but it is not common that the stopes fill themselves, except in the case of orebodies of less width than the full set. The only change from sorting ore underground in the stopes is the use of a picking belt at one property, to segregate the first-class ore.

Of other mining methods, little can be said as to what is generally done. Stopes are in some cases mined by backfilling and also by stulling, but shrinkage has never been a success, because of the character of the walls. For mining caves or old gobs, spiling or booming is common practice, and regular square setting is possible where the gobs have been packed tight enough to hold for short periods of time. Going down on the caves and gob from the top is being successfully done by a method of top slicing, caving the timbers as each floor is mined out and introducing filling from above.

Timbering.—The larger part of the stope timber used in Butte is framed at a large central framing plant near the city. Practically all of the mines have their own timber-framing shops, which are used in most cases to frame miscellaneous jobs, but in a few instances the individual mines frame their own stope timber. Practically no timber is framed underground. Orders are telephoned to the surface and the special work is done as required.

The timber is delivered to the mine yard in railroad cars, and is stored on skids until needed. Timber trucks, pole lagging crates, and mine cars are used to move the various classes of timber about the mine yard. The pole crates and mine cars (used for blocks and wedges) are sent below on cages. Tunnel sets, stope timber, etc. are loaded on the cage decks.

The timber used in drifts is generally round peeled native fir and is of better grade than the average stope timber. In drifting, the tunnel set

is used. Posts are 7 to 8 ft. long and 12 to 18 in. in diameter. The caps are 64 in. over all, 48 in. in the clear, and have a diameter of from 12 to 18 in. Round collar braces 6 in. in diameter are the rule. In some instances, it is advantageous to use tunnel caps reaching from wall to wall. The sets are commonly stood on 64-in. centers, but in mines using other than standard stope timber, the sets are spaced to correspond to the stope, cap to cap dimension. The back is covered with pole lagging, split poles, or 3- or 4-in. plank, varying in width from 8 to 12 in. Pole lagging and split stulls are used on the sides of the drift in heavy ground.

The development and evolution of the square-set framing in Butte has been described.[6] At present, there are two general types of framing. The standard step-down set of round timber 64 by 64 in. by 7¾ ft. center to center, and the single-horn framing. In the latter type of framing, the horn is commonly 5 by 5 by 2½ in., the lengths of posts, caps and girts varying from mine to mine. The latter method is used on sawed and round timber. In using the single horn on round timber, the timber must be of uniform diameter to insure tight-fitting joints. The reason for using the single horn is that great top weight is apt to crush the multiple-set framing. The object of the step-down framing is to provide a good fit for adjacent timbers of greatly varying diameters. Round caps are slabbed 5 in. from the center line to facilitate the laying of floors, which are commonly 3-in. plank cut to center dimension between caps. Auxiliary stope timber consists of blocks, wedges, angle braces, pole lagging, split stulls, and bulkhead material. In rill stopes, long stulls, reaching from wall to wall, are used to hold large slabs of waste.

In silling, selected timber of large diameter is used. Posts are framed on one end only, the framing corresponding to the framing of stope timbers. The posts are 7 ft. 11 in. over all, slight variations from this being made by the various companies. The posts are set directly on the rock floor, as it has been found that mud sills rot out. A sill floor is laid on the rock. The sill drift is covered by sheeting. The sheeting is carried 30 in. above the top of the sill set, and is supported on 10- by 10-in. caps and ties, which rest on pony sets or blocks.

By using the raised sheeting, the drift set may be changed without interfering with motor haulage. The sheeting itself is 4 in. single or double lagging. In two properties, split 6-in. stulls are used. In many cases, when long posts and heavy caps can be used, sheeting may be advantageously eliminated. In veins up to 12 ft. wide, angle sheeting of round timbers is used.

Chute mouths are generally supported on chute sets placed between sill sets. Chute bottoms, inclined 40°, are made of 5-in. plank. Steel turnsheets are used to protect the bottoms in some mines. Chute jaws

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[6] B. H. Dunshee: *Trans.* (1913) **46**, 137.

are made on the surface. When chutes are to be built, an order is placed for a complete chute, which is sent underground ready for erection. Both the arc and flat gates are used with a steel or wooden stop board.

The raises are timbered with sets that correspond to the dimensions of the stope timber used. Raises are usually timbered with square timber, even in mines using round timber in the stopes. Various types of cribbed and bricked raises are employed in heavy ground. Most raises are two compartments, chute and manway; but in rill stopes, three-compartment raises are necessary. Raise sets must be blocked with care to prevent riding and squeezing out of line. Center braces are used to strengthen the set, and the posts must be blocked behind the center braces in heavy ground, or they will be broken. Chutes are lined with 3-in. lagging for the first 100 ft. and with 2-in. above that point. A timber slide is built in the manway compartment to aid in raising and lowering timber and supplies. Landings are built every third set in the manway to comply with the state law. Landings of spaced pole lagging aid in circulation of air.

Timbered stations, with an arched back of 12- by 12-in. timbers, are the tendency. The older stations with flat backs were timbered with 12- by 12-in. to 12- by 16-in. and 14- by 14-in. timber. In heavy moving ground, stations timbered with solid cribbing have proved satisfactory. In one property, concrete stations are constructed on some levels on account of heavy ground.

After the timber is removed from the cage, it is taken directly to the working place, stored in large depots, or in local depots distributed about the level. Timber trucks are used to transport the timber underground. Wedges and blocks are moved in mine cars and pole lagging in crates.

Timber is hoisted into raises by Bacon, Anaconda, and Little Tugger hoists, ¾-in. manila rope and 5/16- to ⅝-in. steel cables being used. A chain and safety hook is attached to the end of the rope for securing the timber. After raises are through to the level above, the timber is lowered into the stopes by means of a rope and snubbing post.

Three companies use timber preservatives. One company treats timber to be used in permanent openings where the conditions are particularly favorable to decay, such as drainage tunnels, upcast shafts, etc. In very heavy ground, it does not pay to treat timber, because the timber crushes before it rots. This company treats such timber with creosote, under pressure, at its framing plant near Butte. Another company has been using Ac-zol where timber is subject to decay. The timber is dipped; no pressure is used. The timber is in the solution but a few hours at most. Another company pickles its timber by submerging and soaking it in copper water. An unused drift or crosscut is dammed up, sprags put in from wall to wall and the timber placed beneath the sprags, so that it will be completely under the copper solution. The tim-

ber is allowed to soak from a few weeks to months, depending on the rate of consumption.

Very little timber is recovered from stopes for reuse. The stope walls must be constantly supported, and there is no opportunity to pull the stope timber. Timber removed in the course of repair work is usually crushed or decayed and is only suitable for bulkheading material. It has been found that timber bulkheads are the best support for heavy moving ground.[7]

Underground Sampling and Check Sampling of Product on Surface, Etc.—The methods of underground sampling have been fully described. There is no check sampling on the surface against specific underground sampling. The nearest approach to this is the mechanical sampling of the railroad cars at the smelter. Every fifth carload is thus sampled and the results listed for each mine. Monthly averages are calculated, then a six months' average is made, and, finally, an average for the year. Of course, these averages are for the mine run as a whole and only serve indirectly as checks on mine sampling and estimates.

Loading Machines and Scrapers.—The introduction of loading machines and scrapers into the district has been slow. The nature of the ore deposits, which are in steeply dipping veins, permits the use of gravity and a large proportion of the ore does not have to be shoveled. Small underground openings caused by the necessary use of timber prevent the use of loading machines of sufficient capacity per man employed to compete with hand methods. Where there has been sufficient ore at one place to warrant the use of loaders, or scrapers, the difficulty has been to supply cars fast enough to keep the loading equipment continuously at work. Various types of portable double tracks, switches, and platforms are being tried to overcome this difficulty. At present there is no standard practice in the use of loaders and scrapers.

Loading machines have been tried by two companies. In both properties, the loader did not prove more economical than hand shoveling. One of these companies is now using a power shovel developed by its mechanical department; this shovel is still in the experimental stage.

Scrapers are being used at two mines of one company and have been used at one mine of another company. The scrapers are of the bottomless type and are pulled by small Tugger hoists. A tail rope is always used.

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[7] B. H. Dunshee: Timbering in the Butte Mines, *Trans.* (1913) **46**, 137.

H. L. Bicknall: Rill Stoping in the Butte District, *Compressed Air Mag.* (July and August, 1921).

N. B. Braly: Shaft Sinking Methods of Butte, *Trans.* (1913) **46**, 151.

Robert Linton: Standardizing by North Butte Mining Co., *Trans.* (1921) **66**, 128.

H. R. Tunnell: Handling Ore in Mines in Butte District, *Trans.* (1922) **68**, 143.

Good results have been obtained at two mines by the use of scrapers on the sill and especially in hot headings near the fire zones.

From the results obtained, it is believed that an ever widening field will be found for the scraper, which it appears is more adapted to the conditions existing in the Butte mines than any of the loaders now on the market.

Tramming and Haulage.—A 4-ton trolley locomotive has been practically recognized as the standard machine, and about 200 of them are in use underground. Most of the locomotives are built for 18-in. gage track, and have the following general over-all dimensions: 36 in. wide, 34 in. high, and approximately 12 ft. long over the bumpers. However, some of the locomotives are built 20-in. gage; these are 2 in. wider.

All of the locomotives have their frames placed outside of the wheels and are equipped with two single-reduction geared motors, hung tandem, and arranged for single-end control. The motors are series wound, totally enclosed, with commutating poles, split frames, rated at approximately 11 hp. each on a 1-hr. basis, and wound to operate on 250–275 volts. They have a rated drawbar pull on a level track of 1500 lb., at a speed of about 6 mi. per hr., and a drawbar pull of 2500 lb. for starting and acceleration.

Each locomotive is equipped with self-locking screw brakes, two headlights, rheostatic control, sanding devices, and a 25-lb. locomotive type bell. The bumpers are designed so that the mine cars cannot climb on top of the locomotive, and are provided with swivel hooks for chain couplings.

Although the operation of the trolley locomotive has been satisfactory, during the past 5 years storage-battery locomotives have been introduced in many mines. Each mining company, however, has its particular type, and there is nothing in common with any one type, except that they all use Edison storage batteries.

The locomotives used by the company that has the greatest number consist of a standard type, weighing 11,000 lb. and having a drawbar pull on a level track of 1500 lb., at a speed of 3½ mi. per hr., and a maximum drawbar pull for starting and accelerating of 2500 lb. The general over-all dimensions of these storage-battery locomotives are: 36 in. wide, 51 in. high, and approximately 12 ft. long over the bumpers.

The locomotive's equipment consists of two 125-volt, ball-bearing, totally enclosed, commutating-pole motors, rated at 13 hp., connected to the driving axles through single-reduction gears. The motors are driven by 105 cells of Edison A5 battery, which are not removed from the locomotive chassis for charging, and have a capacity of 23½ kw.-hr. The speed control is obtained by connecting the motors in various series-parallel combinations through a grid resistance by means of a magnetic blowout, drum-type controller, provided with four series, and three paral-

lel positions. Each locomotive is equipped with an automatic locking-screw type brake, four sand boxes, one 25-lb. locomotive type bell, a circuit breaker, two healidghts, and an ampere-hour meter.

For charging the locomotives, a dead front, safety-first charging panel is used. The general design consists of a steel plate front, equipped with push buttons, rheostat, hand wheel, and a flush-type voltmeter and ammeter. The rheostat and contactors for adjusting and opening or closing the charging circuit are mounted directly back of the panel. This mechanism is enclosed on all sides with protective steel screens, and has a drip-proof steel roof overhead.

The locomotives are charged from the circuits that supply the trolley locomotives and, at each mine, motor-generator sets are placed, either on the surface or underground, for the entire mine haulage system. In some of the mines, a semi-automatic substation is being used; this consists of automatic reclosing circuit breakers.

The tracks throughout the drifts and crosscuts have a uniform grade of 0.5 per cent. toward the shaft in favor of the load, and the curves are laid out on a minimum radius of 18 ft. In those drifts and crosscuts where trolley locomotives are used, a standard 2/0, grooved, trolley wire is strung along the drifts about 6½ ft. above the tracks and 6 in. to one side of the rail, and protected by an inverted wood trough, about 8 in. deep. All tracks where the mining locomotives are used are laid with 25-lb. rail on ties, spaced about 30 in. apart. The average length of rail, however, is about 15 ft., joined by splice bars drilled for four bolts.

A number of different sizes of cars are used throughout the district, but practically all of these cars are of the same general design.

The car known as No. 2 Standard is a popular type, and a large number of them are in use throughout the mines. This car weighs about 1030 lb. It has a capacity of 14 cu. ft. of ore, weighing about 1400 lb., and is equipped with Hyatt roller bearings. It is general practice to haul about ten of these cars per train; they are coupled by three links and a hook between the locomotive and first car, and two links and a hook between cars. The car is rugged and durable and, recently, copper alloy steel is being used in the fabrication of the body.

Underground Storage and Dumping.—Ore is stored underground in cars on stations, in transfer raises, and in skip pockets. The present tendency is toward large stations, with skip pockets beneath. The station size is limited by the character of the ground, moving ground precluding the use of large stations.

The area of stations varies from 500 to 1500 sq. ft. The height is from 8 to 13 ft. All station floors are covered with turnsheets. Cars are stored on stations and on trackage leading from them; the combined capacity averages 30 cars, with a maximum of 70.

The capacity of skip pockets varies from 6 to 280 tons, the average pocket holding 150 tons. Large pockets, with two gates, large centrally

divided pockets for handling ore and waste separately, and the double pockets having one set beneath the other, are used. The common interval between pockets is 200 ft., and in many instances in mines with levels at 100-ft. intervals, transfer raises are used to pass the ore to the skip pocket below. The skip pockets are emptied by opening air gates, and feed directly into skips or through measuring pockets. The latter have hand-operated gates. No automatic pockets are in use. On levels producing small tonnages in mines where skip hoisting is used, cars are dumped directly into skips. This is a makeshift and not standard practice.

Transfer raises are used for storage. The raises are either in the orebody, near the orebody, or else at the shaft. The latter connect the stations to the skip pockets 100 or 200 ft. below. Three types of transfer raises are used. The two- or three-compartment square-set raise, each compartment having an area of 16 sq. ft. and holding 100 tons per 100 ft. of height, the inclined untimbered raise having an area of 30 sq. ft. and holding 190 tons per 100 ft. of height, and the bricked or cribbed raise having an area of 50 sq. ft. and holding 350 tons per 100 ft. of height.

The cars used are end dump, and all cars are dumped by hand. On busy levels, one or two car dumpers are employed, who dump the cars and make up the trains of empties. The cars are dumped through openings at the center of the station, on the sides, or over a long track grizzly running at right angles to the station.

Hoisting.—For the purpose of this paper, hoisting engines on thirty-three main shafts will be considered, of which:

Seventeen are operated by compressed air at 90-lb. gage pressure; ten are operated by steam at from 110- to 135-lb. gage pressure; six are operated electrically.

Sixteen of these shafts are equipped with chippy engines, of which ten are operated by compressed air at 90-lb. gage pressure; four are operated by steam at from 110- to 135-lb. gage pressure; two are operated electrically.

Three of these mines have supply shafts in addition, equipped with engines operated by compressed air at 90-lb. gage pressure.

The main hoists are as follows:

Number................	7
Type and size...........	34 in., 34 by 72 in. Duplex, double-drum, first-motion reversible.
Power.................	Compressed air.
Drums.................	Cast-iron spiders and hub, steel-plate face, straight face, 12-ft. diam. by 56- to 68-in. face.
Ropes.................	1½-in. blue center steel, 6–19 regular or Lang lay, according to conditions.
Brakes.................	Parallel-motion, post-type, wood-lined, gravity operated, released by compressed air.

Clutches................Axial friction-plate type, wood lined, operated by compressed air.
Safeties................Lilly automatic hoist controllers.
Sheaves................Five are 12 ft. in diameter, two are 10 ft. in diameter. All are cast iron with grooves 1/16 in. larger diameter than diameter of rope.
Skips..................One deck and skip each side; deck weighs 3300 to 3700 lb., skip weighs 6500 to 8200 lb.; capacity of skip, six 6 tons, one 7 tons.
Man cages.............Six have separate man cages, four decks each, each side; safety dogs are provided.
Balance................Hoist in balance.

Number................2
Type and size...........34 in., 34 by 72 in. duplex, double-reel, first motion, reversible.
Power.................Compressed air.
Ropes.................7/16 by 7 in. flat.
Brakes.................Parallel-motion post type, wood lined, gravity operated, released by compressed air.
Clutches..............Band type, operated by compressed air.
Safeties................Lilly automatic hoist controllers.
Sheaves................9 ft. 10 in. diameter by 9½ in., cast iron.
Skips..................One deck and skip each side: deck weighs 3900 lb.; skip weighs 8700 lb.; capacity of skip, 6 tons.
Man cages.............Four decks, each side. Safety dogs are provided.
Balance................Hoist in balance.

Number................2.
Type and size...........30 in., 30 by 72 in. duplex, double-reel, first-motion, reversible.
Power.................Steam, 115- to 120-lb. gage pressure.
Ropes.................½ by 7½ in. flat.
Brakes.................Parallel-acting post type, wood lined, gravity operated, released by steam.
Clutches................Band type, steam operated.
Safeties................Lilly automatic hoist controllers.
Sheaves................9 ft. 10 in. diameter by 9½ in., cast iron.
Skips..................One deck and skip each side, deck weighs 3500 lb., skip weighs 9000 lb.; capacity of skip, 6 tons.
Man cages.............Three decks, each side. Safety dogs are provided.
Balance................Hoist in balance.

Number................1.
Type and size...........30 in., 30 by 72 in. duplex, double-drum, first-motion, reversible.
Power.................Steam, 125-lb. gage.
Drums.................Cast-iron spiders and hub, steel-plate face, straight face, 12-ft. diameter by 63-in. face.
Ropes.................1½-in. blue center steel, 6–19, regular lay. Old ropes taken from large electric hoist given several months longer service on this hoist.

Brakes................Parallel-acting post type, wood lined, gravity operated, released by steam.
Clutches................Axial friction-plate type, wood lined, operated by steam.
Safeties................None.
Sheaves................10-ft. diameter, cast-iron hub and rim, bicycle spoke.
Skips................Skip and one deck each side, skip weighs 7500 lb.; capacity of skip, 4.5 tons.
Man cages................Four decks, each side. Safety dogs are provided.
Balance................Hoist in balance.

Number................6.
Type and size................28 in., 28 by 48 in. duplex, double-drum, first-motion, reversible.
Power................Four by compressed air; two by steam; 120-lb. gage pressure.
Drums................Cast-iron, 6-ft. diameter by 56-in. face. Five have straight faces lagged with 3/8-in. plate. One has grooved face.
Ropes................Four use 1 1/4-in. blue center steel, 6–19 regular or Lang lay according to conditions, special construction. Two use 1 1/8-in. blue center steel, 6–19 Lang lay, special construction.
Brakes................Parallel-motion post type, wood lined, gravity operated, released by compressed air or steam.
Clutches................Axial friction plate type, wood lined, operated by compressed air or steam.
Safeties................Lilly automatic hoist controllers.
Sheaves................10-ft. diameter cast iron, with grooves turned 1/16 in. larger diameter than diameter of rope.
Skips................Four have one deck and skip each side: deck weighs 3000 to 3700 lb.; skip weighs 5880 lb.; capacity of skip, 4.1 tons.
Two have three decks each side: top deck weighs 2600 and 3000 lb.; lower decks weigh 1700 lb. and 3000 lb. each; weight of load per car, 2500 lb.
Man cages................Four have separate man cages, three decks each; each side. safety dogs are provided.
Balance................Hoist in balance.

Number................1.
Type and size................27 in., 27 by 72 in. duplex, double-reel, first-motion, reversible.
Power................Steam, 110-lb. gage pressure.
Ropes................1/2 by 7 1/2 in., flat.
Brakes................Post type, pivoted at bottom, and operated at the top, wood lined, gravity operated, released by steam.
Clutches................Friction clutch, steam operated.
Safeties................Lilly automatic hoist controller.
Sheaves................9 ft., 10 in. diameter by 9 1/2 in., cast iron rim, bicycle spoke.
Skips................One deck and skip each side: deck weighs 3500 lb.; skip weighs 9000 lb.; capacity of skip, 6 tons.
Man cages................Four decks; each side. Safety dogs are provided.
Balance................Hoist in balance.

Number................1.
Type and size................23 in., 23 by 60 in. duplex, double-reel, first-motion, reversible.

Power	Compressed air.
Ropes	3/8 by 6 in. flat.
Brakes	Parallel-motion post type, wood lined, gravity operated, released by compressed air.
Clutches	Axial friction-plate type, wood lined, operated by compressed air.
Safeties	Lilly automatic hoist control.
Sheaves	7 ft. 9 in. diameter by 7 in., cast-iron rim, bicycle spoke.
Skips	None.
Cages	Hoist with cages, two decks, each side: Top deck weighs 3600 lb.; lower deck weighs 2000 lb.; load per car weighs 2500 lb. Safety dogs are provided.
Balance	Hoist in balance.

Number	1.
Type and size	22 in., 22 by 48 in. duplex, double-drum, first-motion, reversible.
Power	Steam, 135-lb. gage pressure.
Drums	73-in. diameter by 56-in. face, cast-iron, straight face, face lagged with 1/2-in. plate.
Ropes	1 3/16 in. improved plow steel, 6–17 regular lay.
Brakes	Parallel-motion post type, wood lined, gravity operated, hydraulically released by oil under pressure furnished by an accumulator.
Clutches	Axial friction-plate type, wood lined, hydraulically operated.
Safeties	Lilly automatic hoist controller.
Sheaves	7 ft. 6 in. diameter, cast-iron hub, manganese-steel rim, wrought-iron spokes.
Skips	One deck and skip, each side: deck weighs 3200 lb.; skip weighs 5335 lb.; capacity of skip, 3.3 tons.
Man cages	Three decks; each side. Safety dogs are provided.
Balance	Hoist in balance.

Number	3.
Type and size	20 in., 20 by 48 in. duplex, double-reel, first motion, reversible.
Power	Two operated by steam at 115- to 120-lb. gage pressure; one operated by compressed air.
Ropes	1/2 by 7 in., 1/2 by 5 in., 3/8 by 6 in., flat rope.
Brakes	Parallel-motion post type, wood lined, gravity operated, released by steam or air.
Clutches	Two jaw type, operated by air and steam; one ribbon-band type, operated by steam.
Safeties	Two Lilly automatic hoist controllers; one Welch hoist controller.
Sheaves	Two 6-ft. diameter by 7-in. cast iron. One 7-ft. diameter by 7 1/2-in. cast iron.
Skips	Two, no skips. One, deck and skip, each side: deck weighs 1000 lb.; skip weighs 4000 lb., capacity of skip, 3 tons.

Cages..................Two, two decks each; each side. Top deck weighs 2600 lb.; bottom deck weighs 2000 lb.; total load per car 2500 lb. Safety dogs are provided.
Balance................Hoist in balance.

Number................1.
Type and size...........16 in., 16 by 42 in. duplex, double-reel, first-motion, reversible.
Power.................Compressed air.
Ropes.................⅜ by 4½ in. flat.
Brakes................Post type, pivoted at the bottom and operated at the top, wood lined, gravity operated, released by compressed air.
Clutches...............Jaw type, operated by compressed air.
Safeties................Lilly automatic hoist controllers.
Skips..................None.
Cages..................One deck cage each side: cage weighs 2200 lb., total load one car 2500 lb. Safety dogs are provided.
Balance................Hoist in balance.

Number................1.
Type and size.........18 in., 18 by 36 in. duplex, double-drum, first-motion, reversible.
Power.................Compressed air.
Drums................Cast-iron, 5-ft. diameter by 50-in. face, grooved face. One drum keyed to shaft. One drum loose on shaft.
Ropes.................1 in. blue center steel, 6–19, regular lay.
Brakes................Parallel-motion post type, wood lined, gravity operated, released by compressed air.
Clutch.................Axial friction-plate type on one drum, wood lined, operated by compressed air.
Safeties................None.
Sheaves...............6-ft. diameter cast iron.
Skips..................One deck and skip, each side: decks weigh 1550 lb.; skip weighs 3050 lb., capacity of skip, 2 tons. Safety dogs are provided.
Balance...............Hoist in balance.

Number................1.
Type and size...........14 in., 14 by 36 in. duplex, single-drum, first-motion, reversible.
Power................Steam, 120-lb. gage pressure.
Drum..................Cast iron, 3 ft. 9 in. diameter by 48-in. face, straight face.
Rope..................1 in. blue center steel, 6–19, regular lay.
Brake..................Post type, pivoted at bottom, operated at top, wood lined, gravity operated, released by steam.
Clutch.................None.
Safety.................Welch hoist controller.
Sheave.................6-ft. diameter, cast iron.
Skips..................None.
Cages..................One deck, cage weighs 2570 lb., total load one car 2500 lb. Safety dogs are provided.
Balance................No.

Number................1.
Type and size...........2000 hp., direct-connected, double drum.
Power...................Direct current.
Motor...................2000 hp., shunt-wound, d.c. Westinghouse.
Motor-generator set. . . 2000 hp., 2200-volt motor, direct-connected to 2000-hp. d.c. generator.
69-ton flywheel, laminated iron, 500 r.p.m.
Drums................. 9-ft. diameter by 96-in. face, cast-iron, grooved face.
Ropes..................1½-in. Lang lay, 6–19 and 6–25.
Brakes.................Parallel-motion post type, wood lined, gravity operated, hydraulically released by oil, under pressure, furnished by an accumulator.
Clutches...............Axial friction-plate type, wood line, hydraulically operated.
Safeties...............Ward-Leonard control, Welch automatic hoist controller, Nordberg mechanical control.
Sheaves................12-ft. diameter, cast-iron hub and rim, manganese-steel lined groove, bicycle spokes.
Skips..................One deck and skip, each side: deck weighs 3600 lb., skip weighs 14,000 lb.; capacity of skip, 7 tons.
Man cages..............Four decks, each side. Safety dogs are provided.
Balance................Hoist in balance.

Number................1.
Type and size...........1800-hp. direct-connected, double drum.
Power...................Direct current.
Motor...................1850-hp. shunt-wound, d.c. Westinghouse.
Motor-generator set.....1200-hp., 2200-volt motor direct-connected to 1200-hp. direct-current generator. 55-ton flywheel, laminated iron, 500 r.p.m.
Drums..................12-ft. diameter by 108-in. face, cast-iron, grooved face.
Ropes..................1½-in. blue center steel, 6–19, regular lay.
Brakes.................Parallel-motion post type, wood lined, gravity operated, hydraulically released by oil under pressure furnished by an accumulator. Also band brakes.
Clutches...............Axial friction-plate type, wood lined, hydraulically operated.
Safeties...............Ward-Leonard control, automatic power cut-off, solenoid control for emergency braking at dump.
Sheaves................12-ft. diameter, cast-iron rim and hub, bicycle type spokes, wood-lined groove.
Skips..................One deck and skip, each side: Skip weighs 10,500 lb. capacity skip 7 tons.
Man cages..............Four decks, each side; safety dogs are provided.
Balance................Hoist in balance.

Number................2.
Type and size...........400-hp., geared, double drum.
Power...................2200-volt, three-phase, 60-cycle, a.c.
Motor...................450-hp., 2200-volt, three-phase, 60-cycle, induction. 600 r.p.m.
Drums..................6-ft. diameter by 60-in. face, cast-iron, straight face.
Ropes..................1-in. blue center steel. 6–19, regular lay.

Brakes..... ....Parallel-motion post type, wood lined, gravity operated, hydraulically released by oil under pressure furnished by an accumulator.
Clutches...............Axial friction-plate type, wood lined, hydraulically operated.
Safeties...............Lilly automatic hoist controllers.
Sheaves...............7-ft. diameter and 10-ft. diameter, cast iron.
Skips.................None.
Cages.................Two decks, each side: Top deck weighs 2600 lb.; bottom deck weighs 2000 lb.; load per car 2500 lb.; safety dogs are provided.
Balance...............Hoist in balance.

Number...............1.
Type and size..........1800- hp. 80-r.p.m. direct-connected, double-drum, G. E.
Motor.................1800-hp. shunt-wound, d.c., 225-per cent. momentary overload.
Motor-generator set.....1150-hp., 2200-volt motor, direct connected to 1300-kw. d.c. 525-volt generator.
46-ton flywheel, laminated iron, 514 r.p.m.
Drums.................10-ft. diameter by 96-in., grooved face.
Rope..................1⅜ in. P.S.F.
Brakes................Parallel-motion post type, wood lined, gravity operated, hydraulically released by oil under pressure furnished by an accumulator.
Clutches...............Axial friction-plate type, wood lined, hydraulically operated.
Safeties...............Ward-Leonard control; Nordberg mechanical control.
Weight of.............Skip, cage, rope and ore per side for 3500 ft. depth 34,000 lb., weight of ore 5 tons.
Cages.................Safety dogs are provided.

Number...............1.
Type and size..........112-hp. geared, single-drum, overbalanced.
Power.................440-volt, three-phase, 60-cycle, a.c.
Motor.................112-hp. 440-volt, a.c. induction.
Drum...... ....Single-drum, 4-ft. diameter by 36-in. face, divided in center, straight face.
Rope.......... ......⅞-in. 6–19 regular lay.
Brakes................Parallel-acting post type, hand operated.
Clutch................Band friction clutch for safety, hand operated.
Sheaves...............4-ft. diameter, cast iron; 42-in. diameter on overbalance weight.
Cages.................Two decks: Top deck weighs 2400 lb.; bottom deck weighs 960 lb.; load per car 2500 lb. Safety dogs are provided.

Special overbalance weight rigged so that horsepower is the same lowering two empty cars as it is raising two loaded cars.

The chippy hoists, with one or two exceptions, are of the following general description:

Duplex, first-motion, single straight-face drums, equipped with post brakes, hand or power operated, and most of them are equipped with the Lilly automatic hoist control; 1⅛ and 1¼ in., 6 × 19, regular lay rope is used.

They are the following sizes:

Eight 28 in. 28 by 48 in.
Two 20 in. 20 by 48 in.
One 20 in. 20 by 60 in.
One 18 in. 18 by 36 in.
One 16 in. 16 by 30 in.

The following are also used as chippy hoists:

One 22 in. 22 by 36 in., double drum, first-motion hoist.
One 150-hp. geared, electric, single drum.
One 600-hp. geared, electric, double drum.

Pumping.—Practically all the main pumping in the district is being done by electric plunger pumps of the vertical and horizontal, quintuplex and triplex, geared types. Because of the corrosive action of the water in some of the mines, many of these pumps are fitted with bronze water ends and special acid-resisting plunger. The pumps are operated against heads as high as 1200 ft. Some of the pumps raise the water from large tanks cut around and beneath the stations, and others operate with the suctions flooded. The water columns are made up, in most cases, of 10-ft. lengths of pipe, with flanges shrunk and peened on and are lined with 3/8-in. sheet lead or with 5/8-in. or 3/4-in. sectional wood lining for the corrosive water, and are unlined for the non-corrosive water. They range in size from 6- to 11-in. pipe. For the most part, standard pipe is used for lifts up to 1000 ft. and extra strong pipe for greater lifts.

Twenty-three or more of the main shafts are connected underground on the 2800-ft. level, so this level is used as the main drainage level. The 1200-ft. and the 2200-ft. levels are also used as drainage levels. The mine water is collected in different parts of the workings and conducted from level to level by water boxes in the shafts or by diamond-drill holes, and delivered to the main pumping plants on the most convenient levels, although the greatest quantity is delivered on the 2800-ft. level. Two main pumping plants handle this water to the surface.

The mine water at the other mines is handled by individual pumping plants situated at the different shafts. In most cases, at these other mines, auxiliary equipment of a miscellaneous type is kept on hand for emergencies.

The main pumping units in the district are as follows:

Number	23.
Type and size	Anaconda type, vertical, quintuplex, 7 by 12 in., geared.
Water ends	Bronze.
Plungers	Corrosiron and porcelain shells cemented on to bronze stems.
Capacity	600 gal. per min.

Height lift............600 ft., 1000 ft., 1200 ft.
Motor................Each pump is fitted with two 150-hp. induction motors connected through herringbone gearing on each end of crankshaft, mounted on top of pump.
Power.............2200-volt, 60-cycle, three-phase a.c. and 440-volt, 60-cycle, three phase a.c.
Columns ....10-in. standard pipe lined with 3/8-in. sheet lead, 11-in. standard and extra strong pipe lined with 3/4-in. sectional wood. 10-ft. lengths, flanged.

Number................2.
Type and size..........Aldrich type, vertical, quintuplex, 7 by 12 in., geared. Otherwise same as above.

Number................1.
Type and size...........Deane, vertical triplex, 6½ by 12 in., geared.
Water ends.............Bronze.
Plungers...............Bronze.
Capacity...............300 gal. per min.
Height lift.............600 ft.
Motor................150-hp. induction motor, 440-volt.
Power.................440-volt, 60-cycle, three-phase, a.c.

Number................4.
Type and size...........Aldrich, vertical, quintuplex, 5 by 12 in. geared.
Water ends...........Cast iron.
Plungers...............Cast iron.
Capacity...............300 gal per min.
Height lift.............1000 ft., 1050 ft., 1200 ft.
Motor.................Three have 150-hp., 2200-volt induction motors, connected by flexible coupling to pinion shaft.
One has 125-hp. induction motor, geared.
Power.................2200-volt, 60-cycle, three-phase, a c.
Column................6-in. extra heavy pipe, unlined.

Number................3.
Type and size...........Aldrich, horizontal, quintuplex, 5 by 12 in., geared.
Water ends.............Bronze.
Plungers...............Corrosiron and porcelain shells cemented to wrought-iron stems.
Capacity...............300 gal. per min.
Height lift,.............800 ft., 1000 ft.
Motor.................150-hp. 440-volt induction motor geared to crankshaft.
Power.................440-volt, 60-cycle, three-phase, a.c.
Column................8-in. standard pipe lined with 5/8-in. sectional wood, 10 ft. lengths, flanged.

Number................1.
Type and size...........Aldrich, horizontal, quintuplex, 9 by 12 in., geared.
Water ends.............Bronze.
Plungers...............Bronze.

Capacity...............812 gal. per min.
Height lift.............1200 ft.
Motor.................Two 200-hp., 2200-volt, induction motors geared to each end of crankshaft.
Power..................2200-volt, 60-cycle, three-phase, a.c.
Column................9-in. extra strong pipe lined with 3/4-in. sectional wood 10-ft. lengths, flanged.

Number................1.
Type and size...........Aldrich, vertical, quintuplex, 6½ by 12 in., geared.
Water ends.............Bronze.
Plungers...............Bronze.
Capacity...............300 gal. per min.
Height lift.............600 ft.
Motor.................150-hp., 2200-volt, induction motor geared to end of crankshaft.
Power..................2200-volt, 60-cycle, three-phase, a.c.
Column................9-in. extra strong pipe wood lined.

Number................1.
Type and size...........Vertical triplex, 5 by 10 in., geared.
Water ends.............Cast iron.
Plungers...............Cast iron.
Capacity...............150 gal. per min.
Height lift.............1050 ft.
Motor.................60-hp. induction motor, geared.
Power..................Electric.

Number................1.
Type and size...........Anaconda, vertical, triplex, 5 by 8 in., geared.
Water ends.............Bronze.
Plungers...............Bronze.
Capacity...............150 gal. per min.
Height lift.............800 ft.
Motor.................40-hp., induction motor, geared.
Power..................Electric.
Column................6-in. standard pipe, wood lined, short lengths, flanged.

Number................1.
Type and size...........Aldrich, vertical, triplex, 4 by 9 in.
Water ends.............Cast iron.
Plungers...............Cast iron.

Number................1.
Type and size...........Cameron, 6-in., 10-stage centrifugal pump.
Capacity...............500 gal. per min.
Height lift.............1200 ft.
Motor.................250-hp. induction-motor, 1750 r.p.m., direct connected.
Power..................Electric.
Column................6-in. standard pipe fitted with air chambers and check valves, unlined.

In addition to these, there are a number of various types air and electrically operated pumps used for unwatering small mines, sinking work, and for picking up small quantities of water and delivering into tanks.

Air Compression.—Compression of air for mine and hoisting use in the district is being done almost entirely by electrically driven air compressors of various types and sizes. A number of old cross-compound, duplex, steam-driven compressors have been converted to electric-rope drive by removing the pistons from the steam cylinders or removing the steam cylinders. The pressure generally used is 90-lb. gage at the compressor.

Twenty-one or more of the main mines in the district are operated together, the air being supplied by one central plant assisted by four booster plants at advantageous points. This air is used for hoisting and for mine use underground. Large capacity air receivers are situated at each compressor plant and at each mine served by the system. The air is distributed to the different mines through approximately 15 mi. of surface pipe lines, which vary from 18 to 2 in. in size. Air meters are located at each shaft served by the system, so that an accurate account of the air used underground may be kept and the air properly distributed. The air that serves the main hoists passes through a specially designed sectional surface heater, where it is heated by steam to approximately 300° F. before being used in the hoisting engines.

At the central plant, an hydrostatic system maintains the pressure at or near 90 lb. and provides some storage of air for emergencies. This hydrostatic system consists of a steel tank, 100 ft. in diameter, with a capacity of 500,000 gal. of water, located near the compressor plant and connected through a 42-in. iron pipe to a series of receivers located on the hillside, at an elevation of 208 ft. below the water level in the tank.

The difference in elevation is equivalent to 90-lb. pressure of water. The tank is kept filled to the proper level with water. An 18-in. air line connects the receivers at the building with the tops of the receivers on the hillside and the pressure of the air is equalized against the column of water in the tank and pipe and held fairly constant.

The rest of the mines are served by individual compressor plants located near the different shafts. The main compressor equipment includes the following:

Number................8.
Type and size...........Nordberg high efficiency engine type, 50 in., 30 by 48 in., 75 r.p.m.
Motors.................1200-hp., 2200-volt synchronous motors, 75 r.p.m.
Capacity...............7500 cu. ft. per min. each.

Number................4.
Type and size...........I-R Roegler P.R.E., 2, engine type, 48 by 29 by 36 in., 120 r.p.m.

Motor..................1215-hp., 2200-volt synchronous motors. 120 r.p.m.
Capacity.............7500 cu. ft. per min. each.

Number................2.
Type and size..........Nordberg high-efficiency engine type, 45.5 by 26 by 36 in., one with Corliss valves, one with feather valves. 119 r.p.m.
Motor..................1200-hp., 2200-volt synchronous motors, 119 r.p.m.
Capacity.............7500 cu ft. per min. each.

Number................1.
Type and size..........I-R engine type, 24 by 40 by 27 in., 150 r.p.m.
Motor..................800-hp., 2200-volt synchronous motor, 150 r.p.m.
Capacity..............5750 cu. ft. per min.

Number................1.
Type and size..........I-R Roegler P.R.E. 2, engine type, 38 by 22½ by 27 in., 150 r.p.m.
Motor..................750 hp., 2200-volt synchronous motor, 150 r.p.m.
Capacity..............5257 cu. ft. per min.

Number................1.
Type and size..........Nordberg cross compound, rope-driven, 24 by 40 by 48 in., 73 r.p.m.
Motor..................800-hp., 2200-volt induction motor.
Capacity..............5200 cu. ft. per min.

Number................2.
Type and size..........Ingersoll-Sergeant cross compound, rope-driven, 50¼ by 30¼ by 60 in., 43 r.p.m.
Motor..................One driven by one 800-hp., 2200-volt induction motor.
One driven by three 300-hp., 2200-volt induction motors.
Capacity..............5000 cu. ft. per min., each.

Number................1.
Type and size..........I-R Corliss valve engine type, 36¼ by 20¼ by 30 in., 120 r.p.m.
Motor..................600-hp. 2200-volt synchronous motor, 120 r.p.m.
Capacity..............4000 cu. ft. per min.

Number................2.
Type and size..........Nordberg cross compound, rope-driven, 37 by 22 by 48 in., 65 r.p.m.
Motor..................550-hp., 2200-volt, induction motors.
Capacity..............3500 cu. ft. per min., each.

Number................2.
Type and size ....Rand Drill Co., cross compound, rope-driven, 30¼ by 23¼ by 48 in., 65 r.p.m. converted from steam-driven compressors.

Motor.................600-hp., 2200-volt, induction motors.
Capacity...............3500 cu. ft. per min., each.

Number................3.
Type and size... .. . . Ingersoll-Sergeant cross compound, rope-driven, 38 by 24¼ by 48 in., 61 r.p.m. Converted from steam-driven compressors.
Motor.................600-hp., 2200-volt, induction motors.
Capacity.... ....3500 cu. ft. per min., each.

Number................1.
Type and size...........Ingersoll-Sergeant cross compound, rope-driven, 32¼ by 22¼ by 60 in., 62 r.p.m. Converted from steam-driven compressors.
Motor.................600-hp., 2200-volt, induction motor.
Capacity...............3500 cu. ft. per min.

Number................1.
Type and size...........Nordberg high-efficiency, feather valves, rope-driven, 75 r.p.m.
Motor................ .500-hp., 2200-volt, induction motor, 75 r.p.m.
Capacity....... ......3700 cu. ft. per min.

Number................1.
Type and size....... ...I-R Roegler P.R.E. 2, engine type, 27¼ by 15¼ by 24 in. 150 r.p.m.
Motor.................400-hp., 2200-volt, synchronous motor, 150 r.p.m.
Capacity...............3000 cu ft. per min.

Number................1.
Type and size. ....Nordberg cross compound, belt-driven, 16½ by 28 by 36 in., 92 r.p.m.
Motor.................350-r.p.m., 2200-volt, induction motor.
Capacity...............3000 cu. ft. per min.

Number................2.
Type and size...........I-R Roegler P.R.E. 2, engine type, 29 by 17½ by 21 in., 180 r.p.m.
Motor.................One driven by 375-hp., 2200-volt, synchronous motor. One driven by 435 hp., 2200-volt, synchronous motor.
Capacity...............2800 cu. ft. per min., each.

Number................2.
Type and size...........Ingersoll-Sergeant cross compound, rope-driven, 32¼ by 19¼ by 42 in., 76 r.p.m.
Motor.................500-hp., 2200-volt, induction motor.
Capacity...............2500 cu. ft. per min., each.

Number................1.
Type and size...........Nordberg Corliss valve, rope-driven, 26 by 16 by 36 in., 86 r.p.m.

Motor	300-hp., 2200-volt, induction motor.
Capacity	2265 cu. ft. per min.

Number	4.
Type and size	I-R Imperial type 10, short belt drive, 13 by 23 by 16 in., 150 r.p.m.
Motor	One 150-hp., 2200-volt, induction motor. Three 200-hp., 2200-volt, induction motor.
Capacity	1150 cu. ft., per min., each.

Number	1.
Type and size	Imperial, belt drive, 23 by 13 by 20 in., 106 r.p.m.
Motor	225-hp., 440-volt, induction motor.
Capacity	1000 cu. ft. per min.

Number	1.
Type and size	I-R Imperial type 10-B, short belt drive, 20 by 12 by 16 in., 170 r.p.m.
Motor	150-hp., 220-volt, induction motor.
Capacity	984 cu. ft. per min.

Number	1.
Type and size	I-R Imperial type B-2, short belt drive, 17 by 10 by 14 in., 165 r.p.m.
Motor	125-hp., 2200-volt, induction motor.
Capacity	600 cu. ft. per min.

Number	1.
Type and size	I-R Imperial type 10, short belt drive, 15 by 9 by 12 in., 210 r.p.m.
Motor	75 hp., 2200-volt, induction motor.
Capacity	513 cu. ft. per min.

Number	1.
Type and size	Rand Drill Co., cross compound, steam-driven, steam cylinders, 18- by 34- by 36-in., air cylinders, 20 by 30½ by 36 in.
Capacity	3000 cu. ft. per min.

Ventilation.—Ventilation is, for the greater part, artificial and is generated by main fans situated at the surface, supplemented, in many instances, by underground booster fans. In most mines, the exhaust system is used and the working shafts are the main inlet air courses. In two mines, the pressure system has been adopted. A few of the properties rely on ventilation induced by natural draft.

The most important feature of the ventilation problem is the limitation of the volume of air circulated through the workings that results from the resistance of the deep timbered shafts. To overcome this

difficulty, a lining of concrete slabs has been placed in the timber framing of a number of air shafts. This converts all shaft compartments into separate smooth-surfaced air ducts, and reduces the resistance to the flow of air to about one-quarter that of the shafts with open timber framing.

Similar results have been obtained by cribbed timbering, and by a monolithic reinforced concrete shaft lining. A variation from the typical rectangular shafts is an octagonal shaft with cribbed timbering that is used solely for ventilation purposes. The approximation to circular section results in minimum resistance with economy of excavation and timber.

As a safety precaution, the shaft timber in a number of the downcast working shafts has been fireproofed with gunite, and when present plans are carried out most of the working shafts will be protected in this way.

All surface and underground booster fans are of the modern small-sized type of large volumetric capacity, having high rotative speed. On account of the somewhat unusually high resistance of the deep mine workings, high ventilating pressures or suctions are necessary, so the water-gage pressures developed by the fans range from 3½ to 7½ in. of water. The diameter of fan wheels ranges from 5½ to 10 ft., and the volumes of air drawn from the mines by individual installations are from 70,000 to 250,000 cu. ft. per min. Motor sizes are from 75 to 300 hp. Concrete and steel fireproof construction have been used in the installation of all surface fans (with few unimportant exceptions) and the air ducts from fan to shaft open into the shaft below the collar, so that the surface at the collar is unobstructed and all shaft compartments are open for rescue work or any other purpose.

The general plan of ventilation, for the district as a whole, is that the ventilation of each mine shall be separate and distinct from the adjoining properties. This is realized in reference to main circulation systems, but is modified, to a certain extent, by minor unavoidable movements of air through adjoining dead workings. In the case of one small group of mines, the continuity of veins through these properties has resulted in such a large number of connected workings that, for ventilation purposes, the group must be considered as one mine.

The flow of air is downward through the working shaft, thence through crosscuts and drifts, and, in a majority of cases, upward through stopes to the connections leading to the upcast shaft. On account of irregularity of occurrence of orebodies, ventilation plans are governed by constantly changing requirements, and details of air distribution show considerable variation. An increasing use of underground booster fans is necessary to secure a flow of air through the lower levels, and to divert it to active sections of the mine. The capacity of these fans is from 30,000 to 75,000 cu. ft. of air per minute. They are belt driven by motors of 30 to 75 hp. and the fan wheels range from 4 to 5½ ft. in diameter. Total volume of

air circulated by surface fans is approximately 3,500,000 cu. ft. of air per minute, and this is sufficient to provide more than 500 cu. ft. per minute per man underground.

For the ventilation of dead ends and other places where there is insufficient air circulation, a blower is placed at the nearest available source of fresh air to force the fresh air through canvas pipe to the working face. Pipe equipment for this purpose is 8-in. pipe for raises or other places where there is little room for the pipe, 12-in. pipe for short crosscuts and drifts not over 500 ft. in length, 16-in. pipe for all crosscuts and drifts over 500 ft. in length and for carrying the air into stopes. The blower equipment for 8-in. pipe has a multivane wheel, 15 in. in diameter by 2½ in. wide, and is direct connected to 1740-r.p.m., 3-hp. motor. For 12-in. pipe, the diameter of the blower wheel is 15 in., width of wheel 5 in. and a 1740-r.p.m., 5-hp. motor is used. For 16-in. pipe, the blower has a 24-in. wheel, 6 in. wide, and is driven by a 10 -hp., 1140-r.p.m. motor.

At most of the mines, the inspection of ventilation equipment, such as doors, pipes, and blowers, is constantly made by the ventilation and safety engineer, acting under the direction of the mine foreman. He also assists in working out the most advantageous details of air distribution, and keeps records of the ventilation conditions, making periodical reports to the ventilation engineer.

Lighting.—Electricity is used for lighting all the stations and many of the drifts, though for working in the stopes carbide lamps are used.

The lighting of each station is generally accomplished by three, 100-watt, type C, Mazda lamps. In the drifts, spaced about 100 ft. apart, 50-watt, type B, Mazda lamps are used. Power for this lighting is furnished by a 220/110-volt, three-wire, grounded, neutral system, connected to transformers placed on the surface or at some convenient point on a station underground. It is distributed by means of a three-wire, rubber-covered, lead-jacketed cable, placed in the pump compartment, with taps taken off at each level and run into conduit boxes, which are equipped with the necessary fuses and switches.

For the wiring on the stations, a two-conductor, flat-lay, lead-covered cable, protected by a safety-first switch, is used; whereas in the drifts, two wires supported on porcelain insulators and equipped with weatherproof sockets have been installed. The wiring in the drifts is also protected by fuses and a safety-first switch is cut into the circuit at some convenient point on the stations. No power for lighting is taken from trolley wires, and no lights are provided in the shaft, except at skip pockets.

Telephone and Signaling.—The movement of all mine cages or skips is under the direction of a station tender and, consequently, the despatching of the cages or skips is directly under his supervision.

During the time men or supplies are being distributed to the various levels, the station tender rides the cages. When ore is being hoisted, he is

stationed at the skip pockets. For his use, two distinct signaling systems are installed in each hoisting compartment. One system is known as the A.C. or station bells, because they are operated from the alternating-current power circuits. The other system is called the D.C. or shaft bells, for its source of energy is a set of Edison primary batteries. Placed on the station posts at various levels in the mines, and within easy reach of the men on the cages, are the station and shaft pull switches for both signal systems. The pull switches for the former are equipped with a short rope, whereas the pull switches for the latter have a long rope tied to them, which extends down the shaft about 200 ft. to the second station below.

If the skip pockets are a considerable distance below the station, a pull switch is located at this point, connected to the A.C. signals. If they are only a short distance below, a light cord is dropped down from the station pull switch above to the skip pocket.

In addition to these two systems, there is also installed on each station an independent signaling system, known as the buzzer call. The buzzer system also receives its power supply from the alternating current power circuits.

Set approximately 25 ft. back from the shaft, is the pull switch for the buzzer system. It is placed this distance back from the shaft so that it cannot be mistaken for the main signal switches.

In the event of a power failure, or during the time the shaft is being inspected or repaired, the D.C. pull switches are used to facilitate the sending of signals from a station or any point in the shaft.

Located on one side of the engineer's platform are three single-stroke bells for the three sets of station pull switches. On the other side are three additional bells for the three sets of shaft signals. These bells have different tones, thus enabling the engineer to tell from which shaft or auxiliary compartment the signal is sent. In addition to these six bells, there is a buzzer for the buzzer system.

The electrical connections for the signal system provide that each bell on the engineer's platform be connected in parallel with a lamp, and in series with a double-pole switch. If one of the signal wires or station pull switches becomes short circuited, or grounded, thus causing one of the single stroke bells to be held up, it can readily be disconnected from the circuit. In case of a short circuit, however, the lamp will remain lighted, and indicate when the trouble is removed. The buzzer system is installed in the same manner, except that no light is necessary; a push button is provided on the engineer's platform so that a signal may be sent to each station in the mine, thereby sounding the buzzers on all the stations, so the engineer may signal the station tender.

When a cage is wanted at any station in the mine, the pull switch on the buzzer system is used, and the buzzer on each station is sounded, thus

notifying the station tender that he is wanted at some particular level. The buzzer on the engine platform is also sounded, and the engineer is thereby advised that the station tender is being called; but before the engineer moves the cages, he receives the proper signal, despatched to him by the station tender over either the A.C. or D.C. systems.

Starting from the hoistman's platform, where the signal bells, lights, and a buzzer are located, several cables extend into the mine, and terminate in junction boxes located at the first station. From this point the cables are continued from station to station, passing through junction boxes wherein they are tapped for the pull-switch connections.

The two cables for the main signal system each contain six No. 12 flexible conductors, while the buzzer cable is made up of three No. 8 flexible wires. The individual conductors of the cables are insulated with rubber compound for 600 volts, and have a distinctive cotton braid coloring for distinguishing purposes. The multi cables are protected by a 1/16-in. lead sheath, which is covered by two steel tapes applied in the reverse direction, over which is a layer of hard jute saturated with asphalt and coated with soapstone.

All the cables running between stations are installed in the pump compartment, and are supported by cast-iron cleats bolted to the shaft timbers on 30-ft. centers, except within 300 ft. from the collar of the pump compartment where the cleats are placed 15 ft. apart, so that they will withstand the extra weight caused by the accumulation of ice during the winter.

On stations, each of the cables pass through cast-iron junction boxes, furnished with a hinged door that is held closed against a rubber gasket by wing nuts. The cable entrances are sealed into the junction boxes by a stuffingbox equipped with a rubber gasket. Each wire of the cable in the junction box is connected to brass terminals mounted on an oil slate panel, and each terminal is provided with a bronze spring test link for each wire. The test links provide a ready means for sectionalizing a circuit in case a pull switch should freeze down on the contacts, or a cable become short circuited. For the purpose of keeping the interior of the junction box dry and to indicate an alternating-current power failure, a 10-watt lamp is placed in a standard receptacle back of a boss on the door, which is fitted with green glass.

The main signal cables have six conductors, but only four conductors are used for either the A.C. or D.C. bells. The two extra wires in the A.C. cable are used to supply power for the light that heats the interior of the box, whereas the two extra wires in the D.C. signal cables are used for telephones.

The pull switches on the stations are connected to the circuits in the junction boxes by two No. 12, A.W.G. conductor, flat-laid, lead-covered cables. These conductors also have a distinctive marking.

Many of the mines are provided with a telephone instrument on each station. This instrument is the standard mine telephone in a cast-iron box, equipped with 2500-ohm ringers. Magnets bridged across two conductors in the D.C. signal cable are provided for this purpose.

Records of Unit Production for One Year of Normal Operation

Mining operations in the Butte district are conducted on both the contract and the day-labor systems. One company operates under a bonus system, by which men are paid a bonus when they exceed a predetermined standard for one day's work. Contract work is carried on in all development work, at most mines, on a cubic or linear foot basis, most of the places being by cubic foot. All stopes in most mines are contracted on the cubic-foot basis.

Tons per Man per Hour and Man-hours per Ton

	Tons per Man per Hour	Man-hours per Ton
Miners in stopes (all men in stopes)		
Square set	0.6313	1.5840
Rill (untimbered)	0 8017	1.2473
Rill (timbered)	0.7796	1.2827
Back Filling	0.6106	1.6377
Stull	0 5075	1.9703
Average all stopes	0.6515	1.5349
The foregoing includes sorting.		
Miners on development in ore and waste rock	0.8768	1.1404
All miners	0.6653	1.5039
Underground labor	0.2529	3.9531
Surface labor (ex. office)	2.7292	0.3664
Total organization	0.2280	4.3847

Classification of Labor, Expressed in Percentage of Total.

Ashmen	0.1355	Masons and helpers	0.0279
Boilercleaners	0.1042	Oilers and wipers	0.9032
Boilermakers and helpers	0.3308	Ore loaders	0.0338
Blacksmiths and helpers	0.8975	Ore samplers	0.2441
Cable repairers and helpers	0.4366	Pumpmen	0.9848
Carpenters and helpers	1.2935	Painters	0.0238
Clerks, office	1.6488	Pipemen	0.2962
Diamond-drill men	0.1261	Station tenders	1.9902
Drivers, locomotive	2.2102	Structural ironworkers	0.0245
Drivers, mule or horse	1.0309	Shift bosses	2.4302
Engineers	1.7079	Superintendents and foremen	0.6668
Electricians and helpers	0.4078	Timbermen	19.8592
Firemen	0.5421	Toolmen and boys	1.0471
Foremen's clerk	0.2680	Trammers	0.8698
Laborers	0.9799	Tool sharpeners and helpers	0.4222
Miners	45.1341	Timekeepers	0.7267
Machinists and helpers	0.7163	Watchmen	1.4793

Labor Turnover.—The labor turnover averaged 73 per cent. per month during the past year.

Total Labor Cost Expressed in Percentage of Total Cost of Mining.—The cost of labor represents 68 per cent. of the total cost of mining.

Records of Units of Supplies

Explosives.—The pounds of explosives used per ton of ore produced averages 1.07.

Timber.—The number of board feet of timber, which includes dimension material of all kinds, round timber or stulls, lagging and poles, per ton of ore produced, averages 22.05.

Horsepower.—The horsepower used per month per ton of ore produced is as follows:

Total	0.09248
Mining (compressed air)	0.03181
Haulage and hoisting	0.01879
Pumping	0.01077
Ventilation	0.01057
Lighting	0.00242
Other purposes	0.01812
Cubic feet compressed air	(free air)
For all purposes underground	6,965
For hoisting purposes on surface	9,451

Cost of Supplies Used but Not Taken into Account Above, Expressed in Percentage of Total Cost of Production.—The supplies included under this heading average 4.63 per cent. of the total cost of production.

Total Cost of All Supplies Expressed in Percentage of Total Cost of Mining.—The total cost of all supplies represents 19.38 per cent. of the total cost of mining.

Ore Mined

Under the methods of mining practiced in the Butte District, herein described, the operating companies actually mine 100 per cent. of the commercial ore developed.

On reaching the surface, the ore is automatically dumped from the skip directly into large bins, or it is automatically dumped from the skip into small headframe pockets and from there is trammed a short distance in large cars, by motor, to the large bins. From these bins, the ore is hauled by railroads in 50-ton cars to the reduction works for treatment. No ore is stock-piled.

Safety and Welfare Work

In regard to the safety and welfare work being conducted by the mining companies, it is necessary to separate the two words for the purpose of clarity in description.

Safety Work.—A brief outline of the safety organizations follows:

One company maintains a Bureau of Safety, which consists of a chairman and a general safety committee composed of the heads of the different departments. A safety committee and a safety engineer are provided for each department, while for the mines there is a similar safety committee, and a safety engineer who is also editor of the employees' safety magazine. Each of the operating mines has a safety inspector. All accidents are reported and carefully classified and tabulated, for the purpose of determining the relative seriousness of the various hazards.

The general safety committee does not hold meetings, but as it is composed of the heads of the various operations, the work of furthering safety is done by the individual effort of members of the committee. The mines safety committees meet every two weeks and discuss safety matters, along with others pertaining to the various operating questions. The safety inspectors meet monthly with the safety engineer to discuss accident rates, inspection reports, etc. Safety inspectors spend their entire time at the mine, making daily inspections, report all accidents and, in general, look after the safety and ventilation work at the mines.

This company maintains two large well-equipped mine-rescue stations supplied with 75 two-hour type Paul, 4 two-hour Gibbs, and 16 one-hour Atmos breathing apparatus, and with trained men in attendance at all times. It is part of the safety inspectors' duty to see that the minimum of ten trained men is maintained at each mine, as well as to see that a full quota of first-aid equipment is on hand at all times. This company has about 500 trained apparatus men, including bosses, in its employ.

Another company has a man in charge of the safety and fire prevention work; his title is fire chief. His work is similar to that of the safety inspectors mentioned above. He makes a special fire-condition report monthly to the general manager. This company maintains a safety committee composed of the superintendent, foremen and bosses, to which the fire chief reports unsafe conditions. The general manager calls meetings of this committee as desired, usually about once each month. At these meetings the reports of the fire chief are discussed. The safety meeting is reported by a stenographer and a record made of all action.

This company maintains a rescue station, equipped with 11 two-hour type Paul breathing apparatus and 4 one-hour type Atmos apparatus, recharging station, etc. One trained first-aid man is kept at each shop and at the shaft collars, and all bosses, foremen and members of the engineering staff are compelled to take the mine-rescue training, while such miners and others as may volunteer are also given the training.

Another company holds foremen's meetings every two weeks, at which the reports of the safety and efficiency engineer are discussed. The safety and efficiency engineer makes regular inspections and makes recommendations to the management regarding safety matters. Pro-

vision is made on each level for refuge stations, which consist of sealed dead ends, equipped with compressed-air lines and water, affording a place where men caught in a possible fire may retreat and seal themselves in.[8] Three sets of breathing apparatus are kept at the mine, and a man is assigned to keep them in good order, while all bosses are trained in apparatus work.

The mine is equipped with means for giving stench warning in case of mine fire. Valerianic acid is used in this installation. The shaft collar, and all surface buildings, are equipped with open head sprinklers, supplied by a 4-in. water main. A provision is made for short circuiting the air on the 300-ft. level in order to take care of smoke from possible fire.

Another company maintains a safety engineer, and a safety committee consisting of superintendents, foremen, bosses and first-aid men. This committee meets as a safety committee every two weeks, and as an efficiency committee on alternate weeks. The safety engineer makes regular safety inspections and safety reports, posts safety bulletins, and makes classifications of accidents.

Another company uses a rather unique method in keeping the subject of safety before its men. It employs a safety inspector, who spends most of his time underground taking pictures of unsafe conditions and unsafe practices. These pictures are developed for stereopticon slides and twice a week the entire crew is assembled before tally and these slides shown, with explanations of how to remedy the unsafe condition. Motion pictures are also shown at these meetings. Two-reel comedies are used when available and industrial motion pictures are used at other times. The company is well equipped with mine-rescue apparatus, which is in charge of the safety inspector. Bosses are trained in the use of the apparatus Special safety meetings are held weekly for the bosses. At these meetings, the unsafe practice slides are shown, and the bosses are required to explain the existence of the unsafe condition.

Another company has no safety inspector, but employs an efficiency engineer, who handles matters pertaining to safety, the upkeep of mine-rescue apparatus, training of first-aid and mine-rescue men, and the general safety work of the plant.

General Safety.—All of the mining companies, except one, use the U. S. Bureau of Mines accident classification. This one company uses a classification of its own design, which is slightly more in detail than that of the Bureau of Mines.

Accident rates for all of the companies are fairly uniform. In each case, all accidents, including those that cause no loss of time, are reported and recorded. In all cases, 10,000 shifts is used as a base for accident rate computation, and the total rates range from 12 to 16 per 10,000 shifts. Cases that lose no time, included in this rate, average about 45 per

[8] See erection of barricades during mine fires and after explosions, by J. W. Paul, B. O. Pickard, and M. W. von Bernewitz. Miners' Circular 25, U. S. Bureau of Mines, 1923, 28 pp.

cent. of the total. The serious accident rate per 10,000 shifts ranges from 1.25 to 2.25.

The mining companies coöperate in every possible way, sometimes at considerable expense, in all matters of safety. They hold a miners' safety-first field day annually, each contributing to the expense in proportion to the number of men employed, coöperate with the U. S. Bureau of Mines in its accident prevention work, support the National Safety Council, and other safety organizations.

The operators coöperate fully in helping one another in case of mine fire or other disaster, and at various times have supplied both apparatus and men to operators in other states in cases of mine fire. The Butte fire fighters are reputed to be among the best in the country. It would be difficult, indeed, to find a mine official in Butte who is not vitally interested in the safety of his employees.

Welfare Work.—All of the operating companies take care of sick, disabled, or poverty-stricken employees as far as possible. This work takes the form of charity either in the distribution of money, free fuel, clothing or food, or in furnishing light work to those who are unable longer to perform their usual duties. They also contribute to all the recognized charities, such as the Associated Charities, the Anti-Tuberculosis Society and others and support the Mines League ball teams and all other kinds of legitimate sport; Butte mines band and all other public activities. They maintain a hospital contract, to which employees contribute one dollar per month each, and the companies, through their officials, insist upon employees receiving proper treatment at the hospitals.

In addition, one company established a relief fund with the understanding that for every dollar contributed by the employees, the company would contribute an equivalent amount. This fund is to be used primarily as a loan fund, from which employees may draw for an emergency, with the understanding that repayment of these loans will be made as convenient after the emergency has ceased. The safety-first committee passes on the applications for loans and, if approved, arrangements are made with the treasurer of the fund. The fund is also used for donations to tide over poverty-stricken families or relatives of men who are injured, where there can be no possibility of repayment; the procedure for such donations is the same as in the case of a loan.

Another company carries group insurance. At the end of the first three months of employment, the employee is given, free of cost, a life insurance policy of $500. At the end of one year this is increased to $1000 and is increased annually, thereafter, at the rate of $100 per year until the maximum of $2500 is carried.

Another company maintains an employees' relief organization, or more properly called "Relief Committee." This committee is appointed by the management but the committee elects its own chairman, secretary,

and treasurer. The committee is composed of the heads of departments, foreman, shift bosses, etc. No collections for charity purposes are permitted in the mines, but employees subscribe funds for the relief committee. This committee takes up cases needing assistance, without regard to the length of service, and the company contributes to the fund from time to time as required, the company contribution being one-third to one-half of that of the employees. The company maintains a doctor in addition to its regular hospital contract.

Mining Methods in the Mother Lode District of California

By Stanley L. Arnot, Plymouth, Calif.

(Salt Lake City Meeting, September, 1925)

The Mother Lode district in California is probably better known as the land of Bret Harte and Mark Twain than as a gold-producing district, although in this respect it holds an important place. The history of quartz mining on the Mother Lode dates back to the '50s, when, after the first fever of the gold rush, men began to look for something more permanent, if not more lucrative, than the placers. Since that time, the discoveries, shutdowns, and reopenings of the many mines, together with the varying fortunes of their numerous operators, constitute an interesting chapter in California history, but one too long to include in a description of mining methods.

The district is well situated, so far as physical conditions are concerned. The Mother Lode extends along the western slope of the Sierra Nevada Mountains, about half way between the crest and the Sacramento and San Joaquin Valleys, the country traversed consisting of low rolling hills—typical foot-hills. The elevation varies from 1000 to 1800 ft. and the climate is most desirable. There is an abundance of water, timber is easily obtained from the higher foot-hills a few miles east, and cheap hydro-electric power is generated practically on the Lode. These natural conditions are conducive to low costs in mining and have been the deciding factor in allowing profitable exploitation of the comparatively low-grade orebodies. Transportation facilities are not bad. Five branch railroads, from the main trunk lines in the valley, tap the district at intervals of from 10 to 30 miles, and the roads connecting these are fairly good for trucking purposes.

Labor is made up of several nationalities, the principal ones being American, Spanish, Jugo-Slav, Italian, and Mexican. There were few Mexicans on the Lode prior to 1919, but the shortage of labor, experienced generally by the mining camps of the country in the years following the war, made necessary the hiring of some. The percentage of Mexican labor rapidly increased in 1920 and for a year or so thereafter, but it has now diminished until at most of the mines only a few Mexicans are working. Of the different nationalities, Americans are the most efficient and Mexicans the least.

Geology

The Mother Lode is roughly 90 miles long, and extends from Mariposa County on the south into El Dorado County on the north. Its strike is approximately N. 30° W. and the veins, with few exceptions, have a northeasterly dip. Ransome, in the Mother Lode Folio No. 63, has described the geology of the district very completely. The belt known as the Mother Lode was made a zone of weakness and consequently a path for ascending gold-bearing solutions, or magmas, by major earth movements that took place in both Carboniferous and Jura-Triassic times. The upheaval occurring during Jura-Triassic time was responsible for the present Sierra Nevada mountains and was probably the more important, so far as the Mother Lode is concerned.

The rocks with which the veins are associated are largely slates, schists, and intrusives of andesitic and dioritic character. The veins were doubtless derived from the grano-diorite, which is exposed at the surface east of the Lode. Slates vary from a coal-black clay slate to a highly compressed schistose-looking conglomerate; and schists grade from a soft light-gray mica schist to a dark typical amphibolite schist. The most important intrusive is a meta-andesite locally called greenstone. Contacts between these different rocks are sometimes very distinct; sometimes they consist of transition zones where it is difficult to locate the dividing line.

The Mother Lode may be considered a magnified stringer zone, with the stringers represented by the various veins. Usually not more than two or three of these veins within a given property will be found workable, and in most cases one of these will be found far more important than the others combined. Fissures as a rule are very persistent and, while the quartz filling may alternately pinch and swell, it is very rare, in the case of major veins, that they die out completely. Veins are invariably accompanied by gouges on either or both walls. These gouges vary in thickness from a fraction of an inch up to several feet, and constitute one of the main factors in the choice of mining methods. When the gouge is dry, it gives very little trouble, but when wet it is the bane of the Mother Lode miner. In some cases drifts following gouges several feet thick have been closed tight within a few weeks after drifting had been stopped. Veins occasionally consist of soft crushed quartz, which caves badly; often, when wet, it runs like sand. The walls also are sometimes soft, particularly in the Mariposa slates, and slough badly. Slates usually contain some lime in very thin gouges separating the laminae; when slaked by air, this lime causes swelling in a line perpendicular to the slate cleavage faces. A combination of these three gives little choice in the selection of a mining method. The method adopted must be one where the ore is blocked and mined as quickly as possible thereafter,

keeping the vein and wall supports close to the working face to prevent caving. It is difficult to block out much of an ore reserve in most of the mines, but the tonnage blocked per foot of development is, on the average, large. Veins are of good size and dip at a steep angle from 45° to 80° which is favorable to cheap mining. It is not the cost of breaking, but the cost of supporting the excavation after breaking that is the chief consideration. Veins vary in width from a few inches to 100 ft., but the average would probably be close to 10 ft.

A large percentage of the ore mined consists of quartz, containing free gold and sulfides, the latter averaging about 2 per cent. by weight. In this type of deposit, the free gold represents about 70 per cent. of the recoverable value, and the sulfides, 30 per cent. The principal sulfide is pyrite, with some arsenopyrite and occasionally small amounts of galena. Tellurides of gold have also been found, but in unimportant quantity. Much of the quartz has a banded or ribbon structure; usually this type is of higher grade than the ordinary white quartz. The presence of sulfides in the quartz is usually an indication of good ore, and the presence of galena almost always indicates ore of better than average grade.

Another type of deposit is an enriched amphibolite or chlorite schist, carrying the bulk of the value in free gold and lying adjacent to a quartz vein, the quartz itself being barren. This type of deposit is rare. A more common type consists of what is termed locally "gray ore." This is a highly altered form of the greenstone, usually found at or near a contact with slate, and is accompanied by considerable quartz both in vein form underlying the gray ore and as stringers interlacing the gray ore itself. In this type of deposit, about 90 per cent. of the recoverable value is in the sulfides (chiefly pyrite) and only 10 per cent. in free gold.

Oreshoots, as a rule, are roughly ellipsoidal in form, with the major axis in the line of dip. There is no general rule as to the pitch, or "rake," of the oreshoots but most of them do not depart very far from the line of dip. The dimension along the major axis is usually much greater than the stoping length, which, taken with the lack of "rake," accounts for the fact that hand tramming is the usual method of handling ore at the levels, shafts having been advantageously placed with reference to the orebodies, in most instances.

Exploration

In early days on the Lode, shafts, and, where the topography of the country permitted, adits were driven on the vein. This was the principal method of exploration, although test pits and trenches were used to determine the extent of oreshoots at the surface.

Sinking, drifting, and crosscutting are the methods of exploration now in use. There have been attempts to use the diamond drill but because of the nature of the formations, core recoveries have been low

and the information obtained has been most misleading in some instances. A diamond drill is part of the equipment at one of the mines, where the country rock is a fairly dense schist, but even in this formation, the results obtained have not been entirely satisfactory. Drilling long holes has been tried also, using heavy drifting machines and jointed drill steel, but without success.

Sampling

In later years, for the purpose of determining the length and value of oreshoots at surface, careful sampling of the vein exposed in trenches has been done, but in the past, very little sampling, preliminary to shaft sinking, was attempted. The miner's eye, supplemented by the pan, was largely relied on in the determination of ore.

Sampling, in connection with mine operation, is carried on at most of the mines in a desultory manner. Vein walls are usually clearly defined and the gold content uniformly distributed through the orebody, hence there is little necessity for close sampling in the stopes, as mining progresses. At a few mines, sampling has been given serious consideration. All development work is sampled closely and the results recorded on assay plans. As the veins are often made up of several layers of quartz having different characteristics, samples of these layers are cut separately, and later averaged for the full vein width. At two of the mines, all samples taken are referred to the foot wall, which is called zero, as for example:

Feet		Per Ton
0– 2	Ribbon quartz containing pyrite	$20.00
2– 4	White quartz containing little pyrite	6.00
4– 8	Slate with quartz stringers	3.00
8–10	Sugar quartz and gouge	1.00
	Average 0–10 ft	$6.60

This method of referring the samples to a wall has been found quite helpful in connection with certain geological work as well as in enabling underground bosses to mine more closely.

Satisfactory samples can usually be taken with a hand pick, as the quartz is seldom so hard as to require moiling. The maximum width for any one sample is 5 ft.; the weight varies from 5 to 20 lb. No quartering is attempted underground with the usual daily samples. At most of the mines, the foreman takes the samples. Two mines employ samplers who keep geological data up to date in addition to performing their ordinary duties.

Estimating

Estimating ore on the Mother Lode is usually, though not always, a relatively simple matter because of the regularity of the orebodies.

Cubic contents are computed after blocking out by drifts and raises, and a weight factor applied. Mother Lode ore averages close to 12.5 cu. ft. per ton, in place, and this figure is commonly used. Estimates arrived at in this manner, even when high assays and irregular widths are reduced to the common level, are usually from 10 to 20 per cent. high in value and low in tonnage, depending on the vein width, because of dilution from the soft walls.

A method giving very satisfactory results when, in the case of an operating mine, the ore reserve is required at the end of each year, is the "proportional areas" method. From a vertical longitudinal section of the mine, the areas stoped during the year are found and the tons per square foot as well as the ton-dollars per square foot for each stope, computed from the actual tonnage and value of the ore mined. These figures, obtained from stoped areas above and below an unstoped block of ore, are averaged, and the resulting factors are applied to the unstoped block. This method eliminates the error that would otherwise arise as the result of dilution from the walls, as the estimate conforms with results actually obtained.

Former Mining Methods

Present mining methods differ but little from the systems of early days. In some of the oldest mines, very large timbers were used, some being 36 in. in diameter. In stoping, 16-ft. sets were commonly used, instead of the present standard 8-ft., and, when opening up old workings, the present-day miner, seeing the huge posts and caps, often wonders how the "old-timers" got their timbers into place. Great advances have been made, of course, in the tools for carrying on mining, but the basic principles of the old methods have been found to be sound.

Mine Openings

The principal mine opening on the Mother Lode is the shaft. The earlier shafts were sunk on the vein; but, as this resulted in heavy upkeep costs, later shafts have been sunk in country rock far enough from the vein to insure firm ground. Both vertical and inclined shafts are used, but the greater number are inclined, the slope angle varying fom 45° to 80°. One combination shaft begins at the surface as a vertical and, after passing through the vein, continues as an incline, paralleling the vein about 150 ft. in the foot wall.

Some of the inclines are sunk in the foot-wall country rock, others in the hanging wall, depending on local conditions. In some cases, it has been necessary to sink from one wall into the other because of changes in the country rock with depth.

Where the topography of the country has permitted, adits have been driven as main working entries. Where they are used, it is necessary to

sink internal shafts for deep mining. Most Mother Lode mines, however, are served by shafts.

Shafts vary in size from 6 by 12 ft. to 9 by 18 ft. outside the timbers. Smaller shafts have two compartments, the larger, three. The average size of three-compartment shafts is about 7 by 16 ft., with the compartments measuring 4½ by 5 ft. in the clear. Occasionally the third, or manway, compartment is narrower than the two main hoisting compartments.

Shaft depths vary from a few hundred feet to over 4400 ft. vertically below the surface. Four shafts have reached a depth of nearly 3000 ft. vertically below sea level. The Mother Lode is favored with a low temperature increase with depth, consequently the great depth reached does not affect working conditions to an appreciable extent.

Shaft sets usually consist of 12 by 12 in. timbers spaced 5 ft. apart, framed in the ordinary manner. Large timbers up to 18 by 18 in. are required in some shafts where the ground is heavy. Where the ground gives trouble by swelling, "jacket" sets, bridged out from the main sets, are used. In some of the old shafts, the wall plates consist of round logs up to 3 ft. in diameter. Timber is the only material generally used, although steel sets have been tried—with poor results. One attempt was made to use "gunite," applying it directly to the ground outside the timbers in the hope that sealing off in this manner would stop the swelling, but the result was unsatisfactory. Several shafts are concreted for a short distance below the collar, but none is entirely concreted.

The timber commonly used is local red fir or Oregon pine. Port Orford cedar has been used also; while not so strong as Oregon pine, it has much greater lasting qualities.

Where vertical shafts have been sunk to serve large orebodies, they have paid for themselves through the added economies in hoisting and upkeep.

Adits vary in size from 5 by 7 ft., to 8 by 8 ft. in the clear. The principal one is over a mile in length. Methods of ore handling in adits vary from hand tramming to motor haulage.

Underground Development Plan

The customary procedure followed in opening up an orebody for mining is as follows: The shaft is sunk to a depth sufficient to provide a sump below the level to be opened. A pocket is cut by raising off the shaft, planking nailed to the wall plates usually forming the front of the pocket. Excavation of the station follows, the resulting waste being drawn off from the pocket chutes. A crosscut to the vein is driven off the station, after which drifting proceeds to the limits of the orebody. As soon as drifting has advanced far enough raises are started at advantageous points, and put through to the level above.

The usual interval between levels is 150 ft., which distance has been found to balance the cost of chute upkeep in the stopes with the cost of opening levels. In some cases, conditions justify a closer spacing; in others, a greater interval, up to 250 ft. or more, may be warranted.

The spacing of raises is governed by the shape and size of the orebody. When the orebody is short, one raise in the center of the shoot suffices. When more are necessary, the interval is usually not over 150 ft. Greater spacing increases the cost of handling timbers and other materials in the stopes, as well as the cost of filling from exploratory work at higher levels. After the raises have been connected with the level above, cutting out for the sill floor is started. Chutes are placed at 25-ft. intervals usually, although distances varying from 15 to 50 ft. may be adopted, depending on local conditions.

Intermediate drifts are sometimes established off the raises, but they are resorted to only when the cost of keeping the stope chutes open becomes excessive. Sublevels have not been driven except in a few instances where stoping has been done by the shrinkage system.

Occasionally, where the orebody has been large and the maintenance of the drift expensive, an auxiliary drift in country rock, paralleling the drift on the vein, has been driven to serve as a permanent tramming way. Raises in country rock connect this drift with the stope, tapping it above the filling and being carried up thereafter as ordinary chutes. It has been suggested that the main drifts serving large orebodies be driven in the country rock in the first place, without driving at all in the vein, but this method has not been attempted on the Mother Lode. With this system, chute raises would connect the drift in the foot wall with the vein. The raises would be connected to form the stoping base, or sill floor, and stoping would proceed as usual. The ore below the sill floor would be recovered in stoping from the level below.

In exploratory drifting, it is common practice to carry the drift in the country rock, cross-cutting to the vein at intervals. Such drifts require no timber usually and can be advanced much faster than drifts in the vein.

Details of Shafts, Drifts, and Raises

The manway compartment of the shaft is used to carry the compressed-air main, pump column, power cable, etc., underground. Compressed-air pipes vary in size but are generally either 4 or 6 in. in diameter. Power lines are usually of the three-wire armored-cable type. Sometimes three rubber-covered wires are carried in conduit; occasionally these wires are merely suspended from insulators. The voltage taken underground for power purposes is generally 440, although as high as 2200 volts is in use. Most of the shafts are equipped with some

form of electric-signal system, although in a few cases, even where shafts are very deep, bell-cord pulled by means of levers is used for signaling.

Drifts, as a rule, are 5 by 7 ft. in the clear. The full width of vein, up to 20 ft., is sometimes taken in drifting, but the common practice is to follow the foot wall with a drift of ordinary size, stripping the ore left on the hanging wall side afterward, though in some cases this ore is left to be stoped from below. Compressed-air pipes vary in size from 1 to 4 in., depending on the length of drift and the number of branch air lines connected to the main. The diameter most generally used is 2 in. As the greater number of drifts are designed for hand tramming, 12-lb. rails are commonly used, with 18-in. track gage. Heavier rail is preferred at some mines and track gages up to 22 in. are to be found. Track grades vary from 0.25 to 1.5 per cent., the average being, probably, close to 0.75 per cent. Uniform grades are established when the drifts are in country rock, but when following a vein it is customary to leave the grade to the miner. As a great number of drifts are driven in ground that swells, it would be useless to attempt a uniform grade because of the necessity for continual trackwork even to maintain fair tramming conditions.

Water is conveyed on the levels by ditches at one side of the track, and across openings by means of pipes or wooden boxes. Raises are generally put up on the foot-wall side of the vein, but, as in the case of the drifts, the full width of vein, up to 10 or 12 ft., is often taken. The raise consists of a chute and manway and is usually timbered with ordinary round-timber stope sets placed at 5-ft. centers, thus giving dimensions of about 6 by 12 ft. outside the timbers. Squared timber is sometimes used and, where the ground is more than ordinarily heavy, the sets are 5 ft. high. One method is to carry the chute on the foot-wall side with the manway above it, but this is not common practice.

Ladders and a timber slide are placed in the manway and timbers are handled by means of a small airhoist located at the foot of the raise. Steel is handled in buckets, although sometimes when the raise is serving a stope, the steel is dropped to the drift below through a square, wooden, steel chute.

Drilling and Blasting

For shaft and winze sinking, drills of the plugger type are almost universally used, although in some mines where the rock is quite hard, the drifting type of drill set up on a bar gives better results. With the plugger type drill, hollow hexagonal steel is used. Both ⅞-in. and 1-in. are in use, but the larger size seems to have met with more favor. This type of drill is often used mounted for drifting, in which case water is supplied to the machine; but in sinking, it is usually operated dry.

When drilling in soft to medium ground, the mounted "Jackhamer" gives satisfactory results; but in hard ground, drifting machines of the heavy type are necessary. The latter machines are almost always chucked for 1¼-in. hollow round steel. Water is supplied to the machines by piping from some source giving the required head, or from tanks under air pressure. For stoping and raising, the well-known "jigger" or "widow-maker" is used. On the Mother Lode, dry machines are the rule. Wet machines have been tried, but are not in general use.

The steel used in the stoper type drill is solid, with cruciform section, and is 1 or 1⅛ in. in diameter, the 1 in. being the more common size. Lugs are used on the hollow round and collars on the hexagonal steel. Because of the gougy and "fitchery" nature of the Mother Lode ground, the tappet-chuck type of machine has not been successful.

Several types of bit have been tried, especially the so-called Carr bit, but most of the mines have standardized on the double-taper cross-bit. The usual gage change is ⅛ in. per ft. of run. A longer run or less change per foot of length gives trouble because of the wearing off of the gage.

When raising or sinking, the drill holes are arranged in rows, spaced according to the nature of the ground, and a wedge or "V" cut is taken, usually from the center. When following a vein, as in raising, the holes are inclined to break to the wall being followed. In the stopes, both horizontal and vertical holes are drilled, depending on conditions, and holes are placed to break to a wall or a free face. The bottom cut is generally used in drifting. This cut is especially convenient where the crossbar is used, with drilling and mucking proceeding simultaneously.

Stemming is not generally used although where introduced it has given good results. Even when it is furnished in convenient form, miners will not use stemming if they can avoid it. Ordinary fuse and caps are used for blasting, except in sinking, where the firing is done by electricity.

Very little bulldozing is necessary, as the ore usually breaks into pieces that can be further broken by hammer.

Gelatin dynamite is the usual explosive, in grades of from 30 per cent., for stoping, up to 60 per cent., in tight drifting ground. For development work, 40 per cent. is the strength commonly employed.

Stoping

Several stoping methods have been used on the Mother Lode, including rill, cut-and-fill, stulls and filling, and even shrinkage, but the principal system in use is the square-set and fill.

After the drifts and raises have been driven, enough ground is broken, starting off the raises, to put in the sill timbers and chutes. The sills are heavy timbers reaching from wall to wall, with ample blocking at

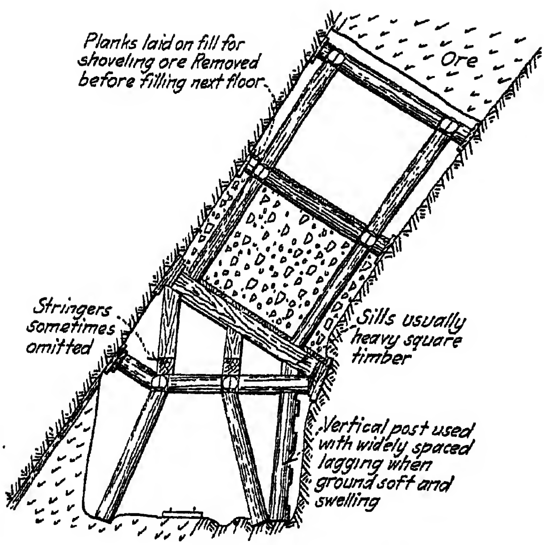

FIG. 1.—INCLINED STOPE SETS USED IN NARROW VEINS.

Before filling begins, the chute sets are lined with planks up to the floor to which filling will be carried; when the vein is wide, a partition

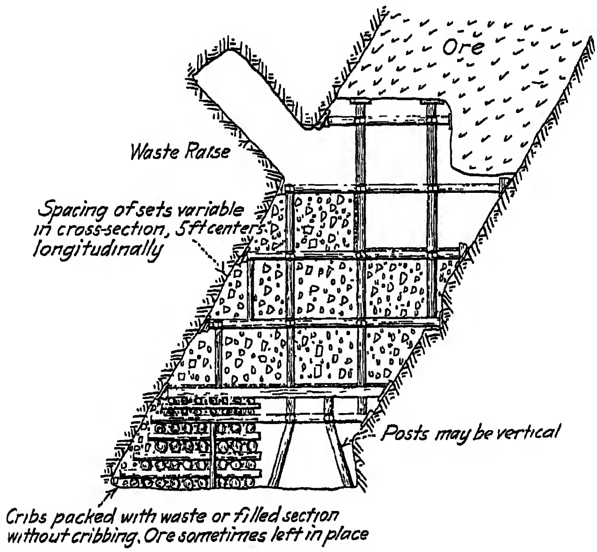

FIG. 2.—SQUARE SETS USED IN WIDE VEINS.

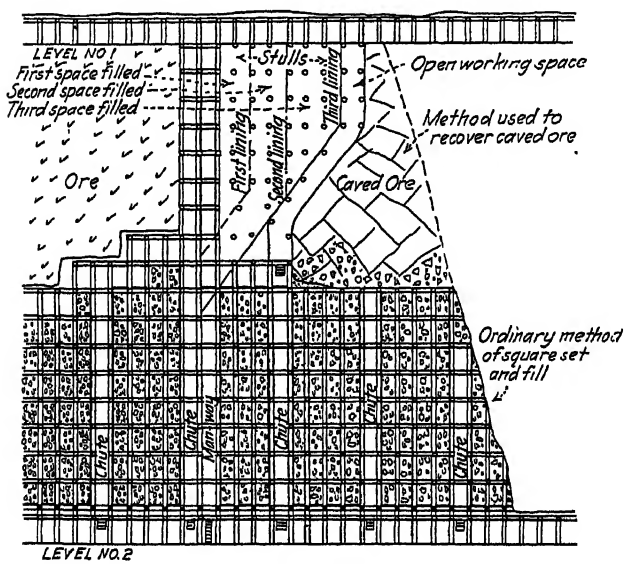

FIG. 3.—LONGITUDINAL SECTION SHOWING TWO METHODS OF STOPING.

is put in, dividing the space between walls; the foot-wall side is generally used for the chute, the hanging-wall side being filled. Cribbing is sometimes used for chute lining. When the filling has reached the desired

floor level, planks are laid down on which to break the ore, which is then shoveled or allowed to flow on slides into the chutes.

Cribs, either built entirely of timbers or filled with waste, are often used to support heavy ground; they are especially useful when gouges become wet and run, allowing large masses of ore to settle. They are also used at the levels to support the hanging-wall side of the sill floor in cases where the full width of vein has been cut out in drifting.

Underhand stoping from the level has been successfully applied to caved blocks of ore, which could not otherwise be recovered economically. Vertical slices of the ore are taken downward, the ore running by gravity to the chute of the raise from which the work is started. Stulls, with posts between, support the walls temporarily. After a considerable area has been opened, lining is nailed to the timbers to leave working room between the lining and the ore and waste is run into the remaining space to form an artificial pillar. Mining is then resumed and the operation repeated until the edge of the ore is reached.

Timber

Native pine is used for stope timber, and native red fir, or Oregon pine, for timbering shafts, drifts, raises, etc. The native pine rots faster and is not so strong as the red fir (locally called spruce) or Oregon pine.

Because of the swelling ground of the Mother Lode, timbers usually break and require replacement long before rotting affects them, hence, while experiments have been made, the use of preservatives has not been given as serious consideration as it has elsewhere.

Round timbers are used almost entirely except in shafts, winzes, or other workings of a permanent nature. In the stopes at some of the mines, squared timber is used for caps, while the posts are round. The average diameter of round timber used is from 12 to 14 in. The usual method of framing is practically the same as that used in the Butte mines. As the wall pressure is generally greater than that resulting from the weight of ore, timbers are framed so that the caps, rather than the posts, butt together. At practically all of the mines, timbers are machine framed at the surface. They are delivered at the shaft stations and stored for use there or at convenient places along the levels. Very little timber is recovered for re-use except the broken timbers removed in repairing drifts, etc.; this timber is generally used for crib building.

Underground Sampling and Check Sampling at Surface

Cut samples are taken from development headings with hand picks, and car samples of the broken ore are taken by the trammers, who remove a handful of ore from each car and deposit it in a marked box conveniently placed. At the end of each day's run of ore, these samples are sacked, numbered, and sent up for assay. Only in a few cases are check samples

taken between the shaft collar and the mill. At the mill, head samples are usually taken by hand, at stated intervals, concentrate samples with a "trier" or by "tenth-shovel" methods, while the tailing sample is obtained by an automatic device, in most instances.

Loading Machines and Scrapers

Loading machines and scrapers are not well suited to most of the Mother Lode mines, but in one mine where some of the stopes are quite flat both loaders and scrapers have been tried. Loading machines have not proved successful but the scrapers have materially reduced ore handling costs.

Tramming and Haulage

Motor haulage on the Mother Lode is rare. One mine handles the ore output with an 8-ton, 550-volt, d.c., trolley-type motor, which hauls seven cars to the trip, each car holding 8 tons of ore. The track grade is 0.25 per cent., the gage 36 in., and 40 lb. rail is used. Where the tramming distance is considerable, hauling is sometimes done by mules. Hand tramming, however, with 18 cu. ft. front dump cars, is the customary method.

Underground Storage and Dumping

Stope chutes and shaft pockets constitute the main underground storage. Shaft pockets are cut on the hanging-wall side, and plank lining attached to the wall-plates forms the front of the bin. Some of the pockets consist of large raises off the shaft with a pillar of rock left between shaft and pocket. Capacities of ore pockets vary from 50 to 300 tons, the average capacity being about 150 tons. Skip loading is done directly from the pocket, by means of a chute gate operated by rack and pinion.

Hoisting

Electrically driven, double-drum, geared hoists are used by practically all the Mother Lode mines, although one of the larger mines is equipped with a steam-driven first-motion hoist. All hoist drums are designed to run either in balance or singly.

Brakes and clutches on the smaller hoists are hand-operated; on the larger, they are actuated by compressed air, steam, or oil under pressure.

Skips are of ordinary type and capacities range from 1 to 4 tons. One mine is equipped with double-decked cages for hoisting and lowering men, but most of the mines use skips for this purpose.

Hoisting speeds range from 500 to 2200 ft. per min., but at most mines, the speed does not exceed 1000 ft. per min. Safety catches are used only in the vertical shafts. Hoisting ropes vary from ⅞ to 1⅜ in.

in diameter, the size most generally used being 1⅛ in. Hoisting is done in balance as much as possible, especially when hoisting ore, although in the course of a day's work, the skips are often run out of balance.

Pumping

Pumping problems do not exist on the Mother Lode, as compared with some other mining districts. The largest mines do not handle over 100,000 gal. of water per 24 hr., and the amount of water made by the mines generally decreases with depth.

Both air and electrically driven pumps are used, but the favorite type for stations seems to be either the vertical or horizontal triplex electric pump. Centrifugal pumps are rarely used. One of the larger mines uses no pumps except in shaft sinking, all the mine water being hoisted in skips to surface.

Air Compression

Compressors of many types and sizes may be found on the Mother Lode. The usual type is the two-stage belt-driven compressor, although the direct-connected type is also in use. Electricity is the only power used at present.

Ventilation

As the Mother Lode mines have become deeper, they have, in most instances, resorted to artificial ventilation. Nearly all the larger mines, having two connections with surface, use exhausters at the collars of the secondary exits. The capacities of these exhausters vary from 30,000 to 60,000 cu. ft. per min. Generally, the main working shaft is the downcast, and the air is led through the workings and up the second exit to the surface. Most of the mines are equipped with doors at the levels for controlling ventilation. Small blowers, using either metal or fabric pipe, are installed underground to meet local ventilation problems.

Lighting

Lighting voltage is stepped down from power lines by underground transformers or, in some cases, taken into the mine after being transformed at surface. The pressure used is generally 110 volts. Usually, only the stations and the shaft at skip loading chutes are electrically lighted, although occasionally lights are placed in crosscuts off the shaft. Individual miners use the carbide lamp, attached to a cap.

Telephone

All mines, except the smallest, use telephones for general communication underground. Telephones are located at stations and at other

points from which it is desired to communicate frequently with other parts of the mine or the surface.

Records of Unit Production

Records of unit production are computed from data obtained at the Plymouth Consolidated Gold Mines Ltd. for the year 1922. The period reported on represents an average year at a typical Mother Lode mine. It was felt that authentic records from one representative mine would be of more value, for the purpose of this paper, than an attempted average, arrived at from data that were difficult to obtain in suitable form, and in some instances not obtainable in any form. It will be readily appreciated that the figures given would not be applicable to some of the Mother Lode mines, but they will represent a fair average for a large percentage where square sets and fill are necessary. Tons per man-hour are obtained by dividing tons per man-shift by eight.

The total production for the period reported on was 92,500 short tons of ore, produced by day labor.

Tons per man per hour (all men in stopes) 0.492; man-hours per ton 2.04.

Tons per man per hour (miners and muckers on development in ore) 0.506; man-hours per ton 1.98.

Tons per man per hour (all underground labor) 0.291; man-hours per ton 3.44.

Tons per man-hour (all surface labor exclusive of office) 1.240; man-hours per ton 0.81.

Tons per man-hour (total organization including office force) 0.225; man-hours per ton 4.44.

Classification of labor expressed in percentage of total:

	Per Cent.
Underground	77.3
Mechanical	9.5
All other surface	8.6
Office	4.6
Total	100.0

Labor turnover varies. The lowest turnover prevails during December and the highest during April. It varies from 9.5 to 61 per cent. per month for all labor; and for underground labor alone, from 15 to 96 per cent. per month. For a few years following the war, the turnover was extremely high.

Total labor cost, expressed as a percentage of total mining cost, for the period under review was 58.2 per cent.

Records of Units of Supplies Used

Records of supply units cover a period of ten years operation ending Dec. 31, 1923.

Explosives, pounds per ton	0.934
Board feet of timber per ton including all timber used, sawed, and round	20.86

Horsepower-hours per ton:

Compressing air	21.20
Hoisting	14.20
Pumping	2.36
Ventilation	1.89
Lighting	0.32
Total	39.97

Cost of all supplies expressed as percentage of total cost of production:

	Per Cent.
Timber	10.0
Explosives	4.0
Power	7.2
All other supplies	11.0
Total	32.2

Ore Mined

Practically all ore estimated is mined, except those blocks of ore that cave so badly as to raise the extraction cost to an uneconomical point.

Disposition of Ore at Surface

After the ore reaches the surface, it is dumped in the ordinary manner into bins. Crushers are located at the shaft in some instances, at the mill in others. After passing the crusher, the ore reaches the mill by hand tramming, mule haulage, motor haulage, belt conveyor, or inclined tramway.

Milling produces two products—amalgam and concentrate. At some of the mines the concentrate is treated locally by cyanidation; others shipped it to the smelter.

Safety and Welfare Work

A central mine-rescue station is supported by the mines and is maintained by a capable foreman, who trains a few men from each mine every two months in the use of mine-rescue apparatus. Two of the mines have complete sets of apparatus and have trained a number of underground

and surface men in their use. Training in first-aid work is given occasionally, under the direction of the Bureau of Mines.

Living conditions on the Mother Lode are good; for which reason little direct welfare work has been undertaken. Most of the mines are situated near small towns where the usual religious and educational facilities, as well as amusements, are to be found; by coöperating with the people of the communities in maintaining and improving these essentials, the mine managements find other forms of welfare work unnecessary.

Mining Methods at the Bunker Hill and Sullivan Mines

By H. M. Childs* and Stanly A. Easton,† Kellogg, Idaho

(New York Meeting, February, 1923)

Methods evolved at Kellogg have had primarily in view the safety of workmen and complete recovery of the ore; it is believed that these results have been secured at a minimum cost. The ore in the Bunker Hill mine is of good grade, relatively speaking; much of it is quite rich and no mining method would be successful which did not recover all of the ore. It would be possible to work certain of the orebodies by shrinkage methods, or some form of top-slicing, at probably lower costs than by the methods now applied, but the shapes of the orebodies are so irregular that satisfactory extraction of the ore in all its ramifications would be extremely difficult.

In past years, much high-grade ore has been lost in blasting, due to the unskilful use of powder in a galena ore, which, by reason of its brittleness and great weight, tends to scatter widely. Any method by which large masses of ore and waste are broken together will result in an imperfect recovery of the ore, and whenever large masses are broken, ore and waste will unavoidably fall down together; ore and waste thus become entangled and, although a pile of broken material may seem to have been well picked over, quantities of ore will inevitably be lost. This fact has been shown by persons reworking the filling in stopes that were mined by methods defective in the foregoing particulars.

Safety of the men is secured by avoiding large openings, and by working mainly under timber, and at small faces at close range and easily inspected. In former years the principal source of serious accidents was "falls of ground;" it is now over seven years since a serious accident has occurred from this cause. The most prolific source of accidents during recent years has been "timber and tools," workmen being struck by timbers passed into the stope from the level above; this danger has been greatly reduced by compressed-air warning whistles and the fencing in of dangerous areas.

The lead-silver ore of the Bunker Hill and Sullivan and affiliated mines is found in bodies of almost every shape, although vertical or nearly

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† General Manager of the same company.

vertical orebodies are unusual. The occurrences may be broadly divided into two general types, the Jersey fissure and the Bunker Hill.

Veins of the Jersey Fissure Type

The Jersey fissure type is represented by a well defined quartz vein ranging in width from a few inches up to 40 ft. and with a dip from 45° to 50°; it carries a sprinkling of siderite but these ores are essentially siliceous as compared with others which carry an excess of iron in the form of carbonate. At intervals, veins of the Jersey type will contain large areas of high-grade, sometimes massive galena. This type of vein has contributed approximately 30 per cent. of the production of the mines.

The veins of this group traverse rather hard quartzite which gives reasonably good walls, permitting the narrow portions of the vein to be worked by stull and filling methods with crib chutes and manways. Ore is always blasted on plank, covered with fine waste to prevent loss of fine ore. The ore is shoveled into wheelbarrows, and coarse waste, if present, is thrown out by a sorter. Walls, parallel to the hanging wall, are built up with the coarsest waste, and other waste is thrown behind; these walls also eliminate the scattering of ore in blasting. Sorting of waste, which in some instances may run as high as 40 per cent. of the broken material, not only enriches the mill feed but furnishes filling for the stope, which is absolutely essential. When the ore has been worked to an inconvenient height above the bottom of the stope, the plank floor is taken up, manways and chutes are raised by additional rounds of cribbing, and enough of the hanging wall is blasted down to fill the stope. In such stopes, enough waste can usually be secured from the hanging wall and from sorting to fill completely all space between walls not occupied by the chutes and manways, which fill a substantial part of the space.

Chutes are spaced 15 to 20 ft. apart, manways 50 ft. apart; they are built with 6-in. half-round cribbing, flat side in, and lined with plank (sometimes steel plates where abrasion is severe). A raise is run from the middle of the stope, following the foot wall, to the level above, serving the stope for the passing of timbers and tools and for ventilation. When the stope reaches half-way to the level above, this raise is used as a passage for the workmen, being easier to travel than through the crib manways. The hanging wall is maintained by stulls and headboards, and sometimes by wall-plates supported by stulls. For widths greater than 40 ft., or for unusually unstable ground, or when the stope arrives within 24 to 30 ft. of the level above, square sets tightly filled with waste replace the stulls. The square sets are used when approaching a level so as to avoid disturbing the track.

Deposits of the Bunker Hill Type

The ore of the Bunker Hill type is found in large irregular masses of galena with siderite and quartz gangue; the galena is often massive. The

practice is to remove all material showing any galena whatever; extremely low-grade material therefore is sometimes moved but, by so doing, high-grade ore is often uncovered. Ordinarily one well defined wall limits the ore; sometimes there is no wall and never two. The dip of the ore ranges around 50°. No two sections of the orebody, even when taken close together, are alike, although there is a rough alignment of their boundaries when considered over substantial distances. The orebodies, in general, range from 300 to 1000 ft. long and from 40 to 125 ft. wide.

The hanging wall is invariably heavy, and close filling with waste is the only method by which it can be safely supported. Square-set timbers are light 12-in. round posts, and 8 by 10-in. caps and ties. These timbers serve merely to prevent slabbing of the rock and to protect the workmen; neither these nor any heavier timbers could support the mass of the hanging wall. The country rock, when permitted to stand uncovered, disintegrates rapidly, but when protected with waste will remain more solid; the practice therefore is to pack waste filling tightly against the hanging wall and bring it up to the apex of the stope as rapidly as possible. Levels are 200 ft. apart and stopes are worked from level to level without transference of ore; this height is probably the limit for efficient use of chutes.

Sampling

No mine sampling is done in the Bunker Hill mine. Everything that shows galena is mined and enough broken waste is sorted out and left in the stope to bring the grade of the broken material up to mill feed quality. The silver ratio is constant so that sampling for silver determination is unnecessary as the galena is a sure marker. The daily samples in duplicate of the mill heads are of the utmost importance in indicating to the mine management the grade of the current mine production, and an effort is made to keep this reasonably uniform and constant. This practice is fairly general throughout all of the Coeur d'Alenes, and the authors know of no property here that has recourse to anything even approaching underground sampling, as it is quite unnecessary.

Stages of Development

The method of developing these orebodies is to crosscut to the ore and then drift on the foot wall to the extremities of the orebody. Fig. 1 shows the Day stope on No. 15 level. When the ore widens so that all cannot be included in the foot-wall drift, other drifts are started on the hanging wall. The entire sill level is finally developed by additional drifts and cutting out.

Fig. 2 and 5, plan and cross-section of the Day stope, show the start of the stoping on the hanging-wall side. A sill floor of plank lagging is laid and square-set timbers placed. These are extended and filled with waste next to the hanging wall, to secure it. A new floor is started above

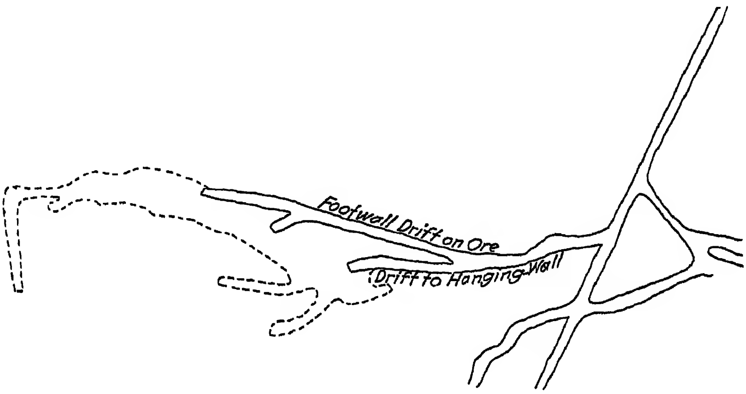


FIG. 1.—DAY STOPE, 15TH LEVEL; FIRST STAGE OF DEVELOPMENT.

the sill floor, the first row of square sets extending into the hanging wall to obtain waste filling for the sill floor, and then toward the foot wall, maintaining a well arched back at all times. This is finished before any stoping is done on the foot-wall side. On the foot-wall drift, at about the

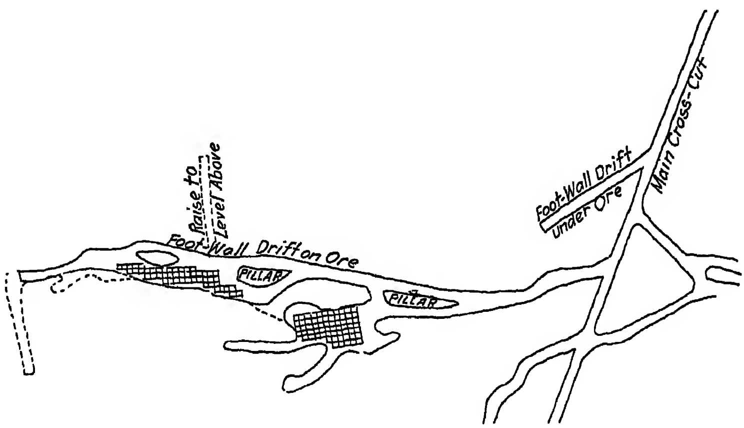


FIG. 2.—SILL FLOOR OF DAY STOPE, AT BEGINNING OF STOPING.

center of the oreshoot, a raise is started, ultimately to reach the level above. Another foot-wall drift in rock is started from the main crosscut, to keep about 20 ft. inside the main foot wall; this is extended at the same time that stoping is begun.

The next stage is shown by Figs. 3 and 6, a plan and cross-section. The foot-wall raise is completed to the level above for passing waste and timbers, and stoping is begun on the foot-wall side of the orebody. The rock foot-wall drift is extended to the end of the orebody. Pillars of ore

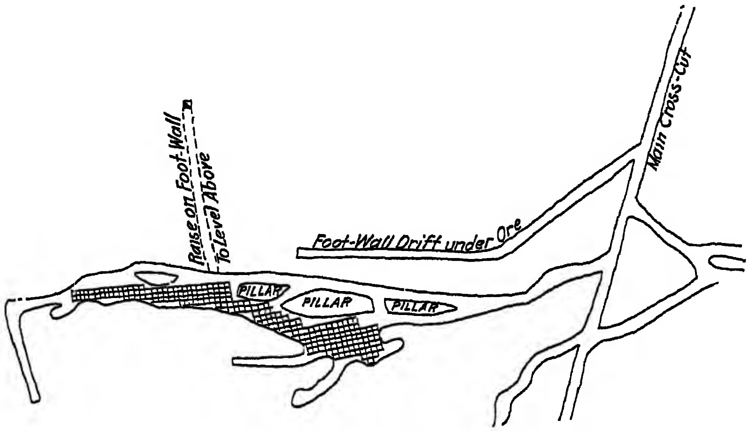


FIG. 3.—PLAN OF DAY STOPE; FOOT-WALL STOPING BEGINNING.

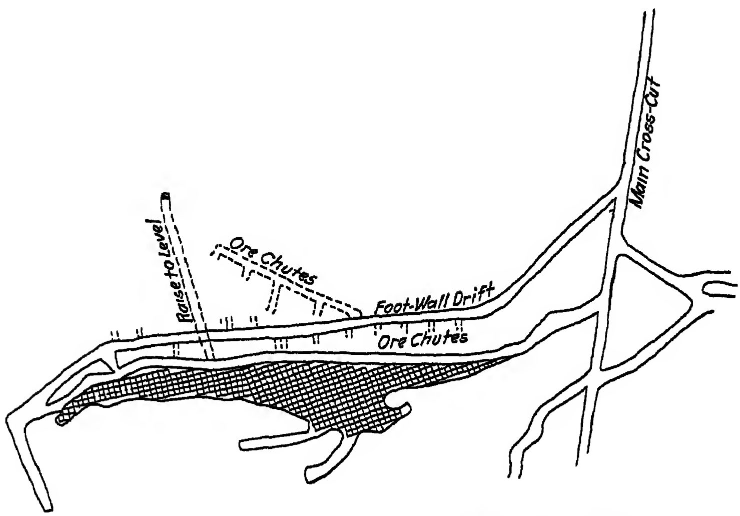


FIG. 4.—DAY STOPE; SILL FLOOR COMPLETELY MINED.

are left on the sill floor to hold the roof, until the advance of the upper filled and arched floors makes it safe to remove them. The square sets are advanced from the hanging-wall side to meet those from the foot-wall side, and filled with waste as they advance. The alignment of these sets, on meeting, may not be straight, but a change of direction is at times

advantageous as thrust is carried to walls instead of by a long line of sets. The main foot-wall drift is generally completed by this time and the boundaries of the orebody have been determined. Drifts are run into hanging wall at several places to make certain of this.

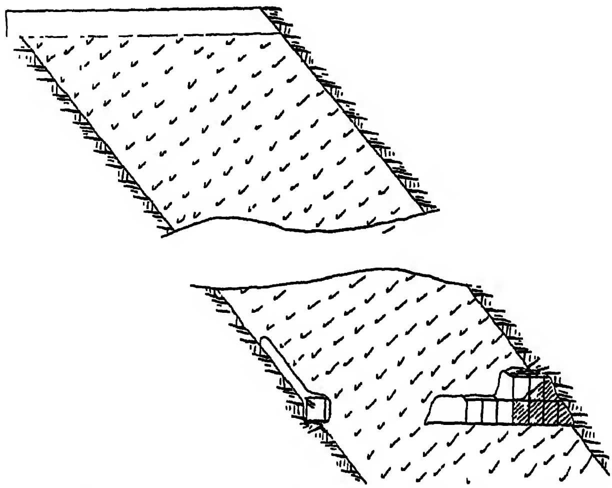


FIG. 5.—SECTION OF DAY STOPE, AT STAGE SHOWN IN FIG. 2.

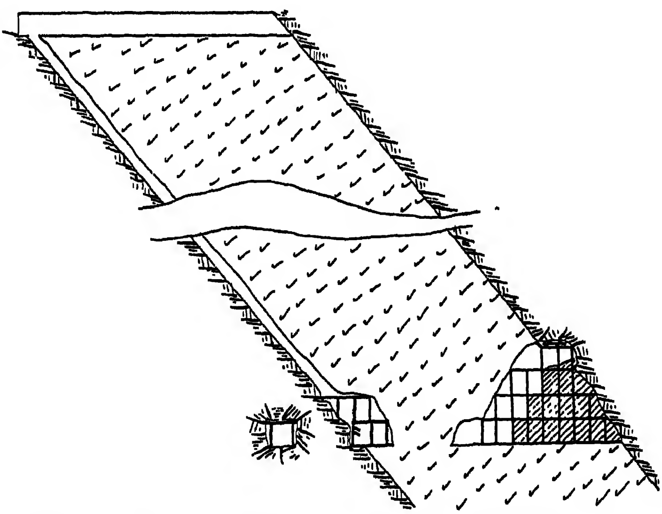


FIG. 6.—SECTION OF DAY STOPE, AT STAGE SHOWN IN FIG. 3.

The complete development of the sill floor is shown by Figs. 4 and 7, a plan and section. Ore chutes are started from the rock drift toward the foot wall of the orebody at intervals of 15 to 20 ft. The hanging-wall portion of the stope is advanced by mining nearly vertical slices, one or

two sets wide; additional floors are put up to keep a strong arch over the stoped area, and a nearly vertical face is maintained. The sill floor is partly filled with waste.

From the top floor, at this stage, a line of sets is extended to the foot-wall raise; track is laid and waste filling is drawn from the raise and run into the stope with scoop cars. Timbers also are brought in by the same route, which likewise becomes the main traveling road as soon as the stope

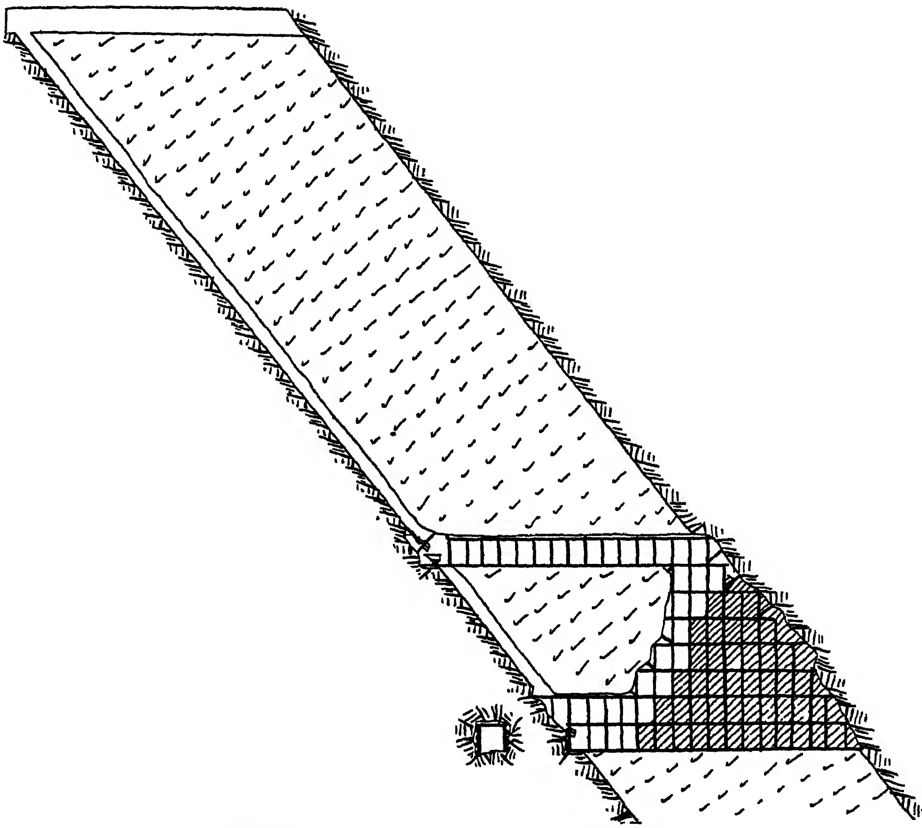


FIG. 7.—SECTION OF DAY STOPE, AT STAGE SHOWN IN FIG. 4.

has reached half-way to the level above. As the stope is here at its highest point and slopes down to each end of the orebody, the timber descends by gravity to the place where it is to be used.

The final abandonment of the sill floor is shown by a cross-section, Fig. 8. On the foot-wall side of the rock drift, a raise is run at a little steeper angle than the foot wall of the ore, and branched at intervals to intersect the ore at the highest floors; it thus serves as an ore chute when these floors advance to the foot wall, and obviates the necessity for transference of ore on an intermediate. If the stope is very long, addi-

tional foot-wall raises, at 200-ft. intervals, are run to the level above, and the stope is developed at each point in the manner already described; such raises generally start from one of the higher floors, from a row of sets run out to the foot wall. Fig. 8 also shows the temporary slides used to get the ore into the chutes and waste in+o the proper pocket.

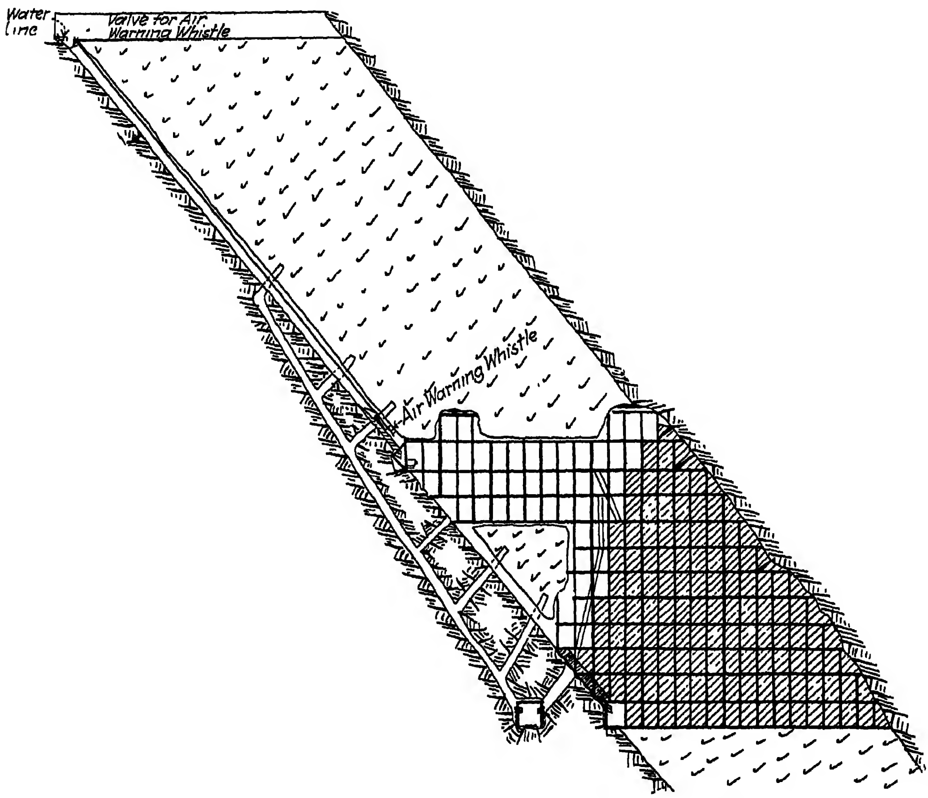


FIG. 8.—SECTION OF STOPE; SILL FLOOR COMPLETELY FILLED.

MINING AND HANDLING OF WASTE

The waste required for filling the stopes is secured from development work in the mine, and also from specially driven raises. These raises are started from levels above those where the waste is to be used so that none of it will require hoisting. A soft or brecciated zone is selected in the foot wall, not near an orebody or any permanent mine opening, and the raise is branched when it comes into this ground. These branches are again branched so that two of them intersect, making a large opening at the junction. When driving the raise, drill holes are put into the back, and these are blasted as soon as the raise is high enough; in this manner, large unsupported openings are made in the brecciated zone, which starts

to cave of itself and continues to do so as long as the waste is drawn out at the bottom. It is drawn into bottom-dump cars of a train pulled by a storage-battery locomotive, and dumped into the top of a foot-wall raise which delivers it to the stope to be filled.

At the bottom of the raise the filling is drawn through a gate into hand-trammed scoop cars. To expedite the passing of the waste down the raise, and also improve its filling and settling qualities, water is often run into the top of the raise until the fine and coarse waste are mixed and have the consistency of wet concrete. When this mixture is run into sets, it consolidates into a dense mass and makes a much better fill than if introduced dry.

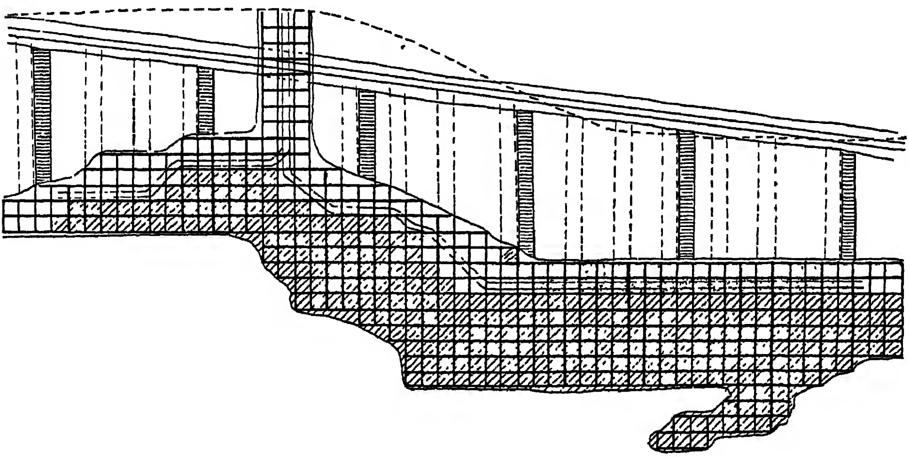


FIG. 9.—PLAN OF TOP FLOOR OF STOPE, SHOWING WASTE DISTRIBUTION.

The foot-wall raise is divided into a waste chute, a timber chute, and a manway, although the waste and timber chute may be combined into one chute. At the foot of the timber chute is a compressed-air whistle operated by a valve at the top of the raise, which blows continuously when timber is going down.

Fig. 9 shows a plan of a top floor of the Day stope, illustrating the method of distributing waste. The sets containing dots have been filled below this floor. In case of a heavy stope, waste is run into these lagged-up sets, up to the ore face, except a set over a chute, which is left open. From this open set, one set is cut out toward the foot wall and is then worked, one set wide, either up or down, and parallel to the walls of the stope. Filled and open sections alternate, so that at no time is a stope left unsupported by filling at the face. Fig. 9 shows the rock chutes 15 ft. apart, reaching to the foot wall of the stope, and the cribbed manways 50 ft. apart.

Advantages of the Present System

Fig. 8 illustrates the advantages of a vertical working face, one of which is that drilling of the ore by uppers is much easier and quicker than by wet, flat holes. Mucking is saved, as the ore falls nearly vertically into the chutes, where sorting is unnecessary. The saving of mineral is large, as the tightly lagged and filled sets act as a buffer for the blasted ore, and the ore can be easily cleaned up from the board floors, further-

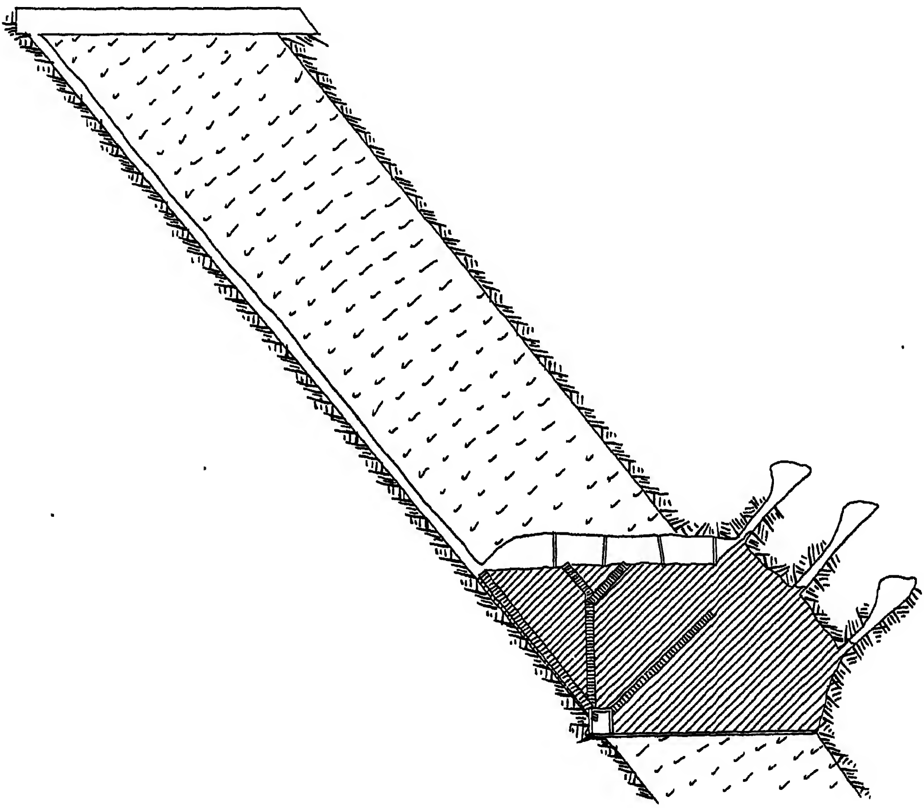


Fig. 10.—Section of Becker stope; stulled and filled.

more, the lagged and filled sets being so close to the ore face leaves no large space over which the blasted ore can be scattered. Sorting of waste out of the ore is done in the same manner as in the small stulled stopes.

The straight, roomy, and solid drifts in the foot-wall rock are much superior to the usual stope drift in ore, and storage-battery locomotives have many advantages, under our operating conditions, over trolley locomotives. At the shaft stations, a 50° incline, 75 to 100 ft. long, is run up from the shaft to the floor of the station, and divided into

two compartments. The trains drop their ore from bottom-dump cars into these pockets very quickly, as uncoupling of cars is not necessary. Flow of ore to the skip is controlled by over-cut arc gates.

Fig. 10 illustrates a section of the old stull and filled Becker stope, with its flat back and cribbed chutes through the filling. Lack of arching

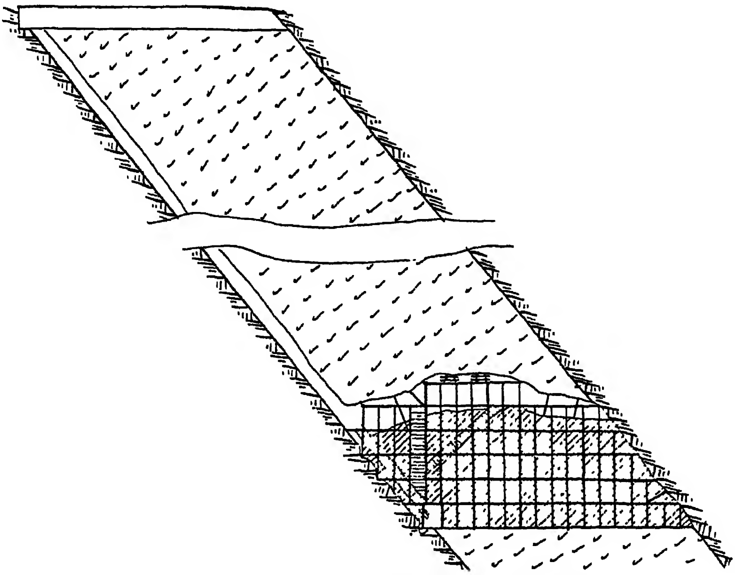


FIG. 11.—OLD TYPE OF SQUARE-SET STOPE.

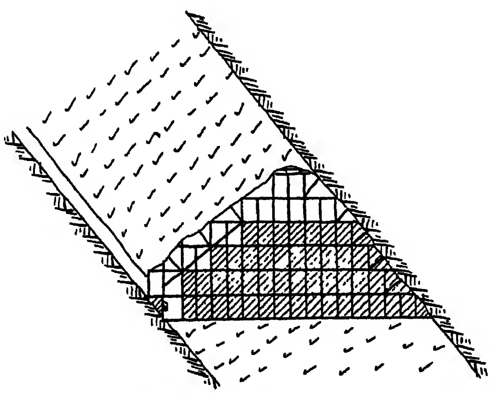


FIG. 12.—TRANSITION FROM SQUARE-SET TO ARCHED-BACK STOPE.

permits falls of ground, as it is impossible to hold it with stulls that are footed on the loose filling. When these falls occur it is impossible to recover all the ore. Even though planking and fine waste are spread before blasting, the ore is scattered over a large chamber and much of it is lost.

Waste for filling this old type of stope was secured by running waste raises into the hanging wall; but as a number of them had to be driven as the stope worked upward, the hanging wall became insecure. The cribbed chutes through the filling were often lost by crushing, or became useless by running into the hanging wall; in that case, branching became necessary, with the usual trouble at the bend.

Fig. 11 is a cross-section of the old type of square-set stope. This is better than the old stull and filled stope, but still lacks the advantages of an arched roof. Fig. 12 shows a stage in the transition from the old square-set stope to the present nearly vertical face of ore.

Summary of Mining Practice

The essential features of the present mining practice, and its advantages, are these:

1. Waste filling, secured from: (*a*) Sorting; (*b*) barren portions of stope; (*c*) caving in waste raises run from next higher level.
2. Waste filling easily placed, mainly by gravity.
3. Light square-sets, closely floored and lagged to prevent waste of ore by scattering when blasted.
4. Prompt placing of filling against hanging wall to prevent air slacking.
5. Vertical, or nearly vertical working faces of ore, resulting in easy breaking of ore by uppers, and easy descent of broken ore into chutes with scarcely any shoveling.
6. Ground is cut first along hanging wall limit of orebody, which is always worked as the highest limit of the stope.
7. The sharp arching of the back resulting from the above procedure greatly increases the strength of the ground; this arch is developed early in the stoping operation and is carefully preserved until the stope reaches the level above.
8. Ventilation is good, since air currents rise more readily through vertical openings than through large horizontal chambers.
9. Haulage drift in the foot-wall rock, connected to the stope by rock chutes, is free from the danger of settling.
10. The large space provided by the rock chutes gives good storage for broken ore; they also avoid the necessity for intermediate handling, even in orebodies having less than the usual slope.

Unit Production for Year 1922

The total production for the year was 421,532 short tons, by day labor.

LABOR AND TONNAGE

	Man-hours per Year	Tonnage	Tons per Man-hour	Man-hours per Ton
Miners in stopes (all men in stopes)	792,642	419,533	0.529	1.889
Miners on development in ore	35,048	19,999	0 570	1.752
Miners on development in rock	5,232	1,593	0.304	3.289
All miners	252,968	421,532	1.666	0.600
Underground labor	948,000	421,532	0.444	2.248
Surface labor (exclusive of office)	155,960	421,532	2.702	0.369
Office labor	72,000			
Total for organization	1,175,960	421,532	0.358	2.789

Classification of labor, expressed in percentage of total, was:

Total Labor	Man-hours	Per Cent.
Mine	948,000	80.6
Surface	227,960	19.4
Total	1,175,960	100.0

The total number of men employed was 567; a normal crew numbered 420 to 450 men.

The total labor cost was $514,500 and the total mining cost $1,180,-693, hence the labor cost was 43.5 per cent. of total cost of mining.

SUPPLIES

(For 421,532 tons produced)

Explosives:

	Quantity	Per Ton
Powder	239,250 lb.	0.567
Caps	1,404 boxes	0.0033
Fuse	924,000 ft.	2.192
Timber	6,435,980 ft. B.M.	15.26

Yearly horsepower-hours = 18,790,071; 365 days = 8760 hr.

$$\frac{18,790,071}{8760} = 2145 \text{ hp.}; \frac{2145 \text{ hp.}}{421,532 \text{ tons}} = 0.005 \text{ hp. per ton}$$

Compressed air = 5,417,421 hp.-hr.

$$\frac{5,417,421}{8760} = 618.4 \text{ hp.}, \frac{618.4}{421,532} = 0.0014 \text{ hp. per ton}$$

Haulage = 1,138,838 hp.-hr.

$$\frac{1,138,838}{8760} = 130 \text{ hp.}; \frac{130}{421,532} = 0.0003 \text{ hp. per ton}$$

Hoisting = 7,217,714 hp.-hr. (tons hoisted = 381,795).

$$\frac{7,217,714}{8760} = 824 \text{ hp.}; \frac{824}{381,795} = 0.002 \text{ hp. per ton}$$

Pumping = 3,353,584 hp.-hr.

$$\frac{3,353,584}{8760} = 383 \text{ hp.}; \frac{383}{421,532} = 0.0009 \text{ hp. per ton}$$

Ventilation = 1,019,176 hp.-hr.

$$\frac{1,019,176}{8760} = 116.34 \text{ hp.}; \frac{116.34}{421,532} = 0.00027 \text{ hp. per ton}$$

Lighting = 643,690 hp.-hr.

$$\frac{643,690}{8760} = 73.48 \text{ hp.}; \frac{73.48}{421,532} = 0.00017 \text{ hp. per ton}$$

Compressed air = 27,077,313 cu. ft.

$$\frac{27,077,313}{421,532} = 64.235 \text{ cu. ft. per ton}$$

The cost of supplies used but not taken into account in the above, was as follows:

Illuminants	$ 3,864
Miscellaneous supplies	66,447
	$70,311

Total cost of production:	
Labor	$514,500
Supplies	229,361
	$743,861

Mining Methods of Hecla Mining Co.

By James F. McCarthy, Wallace, Ida., and Charles H. Foreman, Burke, Ida.

(New York Meeting, February, 1923)

The orebodies of the Hecla mine are from 3 to 40 ft. wide, dip not less than 70°, and in most cases are nearly vertical. The Hecla and Intermediate orebodies are generally associated with a lamprophyric dike, usually about 2 ft. wide, but which attains a maximum width of 7 ft. The ore appears on either or both sides of the dike. The country rock is Burke quartzite with no definite walls and a tendency for nearly horizontal, but irregular, quartz stringers to run from the lode into the walls. This latter condition prevails almost entirely in the hanging wall, causing a natural tendency for portions of the wall to "offset" and resulting in an extremely heavy side pressure. The fact that there are no definite walls causes a great variation in the width of the stopes, as stringers will run into the walls for irregular distances. The ore occurs as galena in varying widths and grades. It is often necessary to mine a face having a "horse" of waste between two stringers of ore. This block of waste might be 6 or 7 ft. wide but with the Hecla method it is necessary to mine the entire face. In this case an effort is made only to shatter the waste, allowing for easy sorting underground.

The Hecla mine requires a flexible system allowing for the extraction of ore from varying widths, with the necessity of close filling to give the needed protection from side weight. About 15 per cent. of the material broken in the face is sorted as waste, either at the face of stope, or in the sorting plant.

Shafts, Drifts and Crosscuts

The outside dimensions of the shaft are 19 ft. 4 in. by 7 ft. 2 in. The shaft consists of three working compartments, two for the hoisting of ore and one for auxiliary work. The working compartments are 5 ft. 2 in. by 4 ft. 4 in. inside measurement. The rest of the shaft is used for pump column, electric cables, and ladderway. (See Fig. 6 for framing of timbers.)

The shaft is about 700 ft. from the orebody for topographical reasons. Solid formation was not reached in shaft for 80 ft. Timbers in this section became alternately wet and dry on account of seasonal changes

and rotted so the shaft was concreted for this distance. Bottom of shaft is 2212.5 ft. below collar. (See Fig. 7 for details of 2000-ft. station skip pockets.)

Main haulage drifts are driven 8 ft. wide by 8 ft. high, both measurements being clear of the timbers. When ore is encountered in the drifts, the width is increased to the width of the ledge, up to 15 ft. The distance from rail to bottom of cap is also increased to 9 ft., to allow handling timbers in timber slides.

Star crosscut is being driven 8 by 8 ft. in the clear. Actual cross-section of this crosscut during June was 81 sq. ft., which allows for irregularities in the walls. General prospecting crosscuts are driven 5 by 7 ft. in the clear.

Underground Development Plans

General plan of development on the Hecla ledge is as follows: Sink shaft below the proposed level to give sufficient room for loading skip from skip pocket and for storage of water in shaft sump below this loading point. The station, skip pocket, and station sump are cut on the new level and the drift started for orebody. After orebody is encountered, a raise is started to the level above, when drift is advanced to a point beyond the raise sufficient to prevent interference with work in raise. Raises are driven from 200 to 300 ft. apart, depending on the character of the orebody above.

The mine was originally equipped with 18-in. gage tracks; at present all levels except one are equipped with 24-in. gage. Main haulage levels use 35-lb. rail. Star crosscut is fitted with 45-lb. rail. Prospecting crosscuts entered by electric motor use 20-lb. rail; 16- and 20-lb. rail are used for tramming in stopes and shorter crosscuts.

Water is conveyed, on levels, by a ditch alongside the track. When the level passes over old stopes, wooden flumes are used as a rule, though 6-in. pipe is used in some cases.

Compressed air is taken down shaft in 6-in. pipe and is connected with receivers on the levels. In the main drifts, 4-in. air pipe is used; 1¼-in. pipe is used in raises, manways, and in stopes. In Star crosscut 6-in. air pipe is used.

In blind drifts and crosscuts, 10- and 12-in. ventilating pipe is used; in Star crosscut, 22⅝-in. pipe.

Mining

Drilling and Blasting

For drifting, crosscutting and stoping, the Ingersoll-Leyner No. 248 machine is used; 1¼-in. round hollow drill steel is made into a cross bit of the following sizes:

Diameter of Bit, Inches	Length of Steel, Feet	Diameter of Bit, Inches	Length of Steel, Feet
2¼	3	1¾	7
2⅛	4	1⅝	8
2	5	1½	9
1⅞	6		

The weight of the Ingersoll-Leyner No. 248 is about 160 lb. For raises and some stoping, the Ingersoll CCW11 wet stoper is used; this machine weighs 93 lb. Drill steel is made from 1-in. hollow steel, a cross bit being used. Drill steels for this machine are made to the following dimensions:

Diameter of Bit, Inches	Length of Steel, Feet	Length of Steel, Inches	Diameter of Bit, Inches	Length of Steel, Feet	Length of Steel, Inches
2¼	3		1¾	8	0
2⅛	4	3	1⅝	9	3
2	5	6	1½	10	6
1⅞	6	9			

The Jackhamer BCRW 430 is used in drilling boulders for blasting in the stopes. For this work, a 1⅝-in. bit is used, the length of steel being from 2 to 3 ft. Steel is made from ⅞-in. hollow hexagon.

The mounted Jackhamer has been used for crosscutting in isolated places. For this work, steel was made up in the following dimensions:

Diameter of Bit, Inches	Length of Steel, Feet
1¾	3
1⅝	4
1½	5
1⅜	6

Two men drove a 5- by 7-ft. crosscut with this machine, averaging 4 ft. per round (round every shift). The weight of the Jackhamer unmounted is 40 lb. During the last shaft sinking, the BCR53 Bull Moose Jackhamer was used; weight 90 lb. Steel is made from 1⅛-in. hollow hexagon.

Diameter of Bit, Inches	Length of Steel, Feet	Length of Steel, Inches
2¼	3	0
2⅛	4	3
2	5	6
1⅞	6	9
1¾	8	0
1⅝	9	3
1½	10	6

All of the steels are made up and sharpened on the Ingersoll-Leyner drill sharpener I-R-5.

The average crosscut is broken with twelve holes; *i.e.* 3 back holes, 3 breast holes, 3 cut holes, and 3 lifters. In tight ground, 3 "second cut" holes are added. In the Star crosscut, four of each of the holes are used.

The number of rows of holes in a drift depends on the width, about one row for each 30 in. Back holes incline about plus 20° from the horizontal: breast holes run horizontal. First cut holes average about 30° below the horizontal, second cuts about 45°. Lifters are inclined about 10° below the horizontal and are bottomed about 6 in. below top of tie.

In stoping, only the back, breast, and cut holes are used, being placed in a vertical row. One row used for each 30 in. of face.

In drifting or crosscutting, the holes are blasted in the following order: second cuts, first cuts, breast, back holes, and lifters. In breast stoping, the cut holes are fired first, then the breast, and then the back holes. In drifting in hard ground, two easers are added to the round being drilled from the same set up as the lifters, but over the bar. When easers are drilled, they are blasted after the second cut holes.

The miner's rule for setting up the machine cross bar is to place it one pick handle from the back and from face.

Powder

The powder used is 1⅛- by 8-in. gelatine, 30 per cent. being used in the stopes and 40 per cent. in the drifts and crosscuts. In the Star crosscut a small amount of 1¼-in. 60 per cent. gelatine is used in the bottom of the cut holes.

Only 7-X caps are used.

Drifting and Stoping

Mine levels are driven from 200 to 400 ft. apart. The drifts are driven the full width of the vein up to 15 ft. When the ledge is wider than 15 ft., the first floor is taken out for the same width, the cap of this set being the full width of the ledge. The bottom of the waste fill is placed on this cap. Ore left in the walls in this manner is recovered when stoping from the level below.

The system now used in the Hecla and Intermediate orebodies may be termed horizontal stoping with stull sets on waste fill. Timber rills and untimbered rills have been used in certain parts but were found not adaptable.

Shrinkage stopes have been used in the East orebody and Ore-No-Go orebody but the method has been practically abandoned.

The stopes in the Hecla have shown a tendency to widen with depth, causing numerous changes in detail to meet new conditions. The early method of placing a chute, manway, and timber slide every 50 ft. (center to center of chute) was abandoned because of excessive side weight, causing a large amount of repairing in the raises and manways. With this method, only about 70 per cent. of the excavation was filled with waste.

The raises were then placed 100 ft. apart, a track laid on the waste fill, and three floors mined out before filling. All of the broken material was dropped to the "muck" floor, directly over the track floor. A system of crossboards on this floor allowed the broken ore to be drawn through the floor into a car on the track floor, to be trammed to the chute. Filling was delayed in this case until the ore in the section was mined for four floors above the track. After mining, the track was raised three floors, and was used to fill the section with waste. No mining was done in the section while filling was in progress. This made the production of any one stope intermittent, causing an irregular flow of ore to the mill. This fact, together with the gradual widening of the stopes with depth, caused variations in this method, cutting down the number of floors removed before filling.

In the present horizontal stoping method, the stopes are started from the raise and one floor advanced four sets and timbered. Work is then started on the floor above for the same distance, and timbered to the face. Further advance is made by setting up the machine on the top floor and drilling the face for both floors. The ideal method of starting the stope would be on a floor on the top of the waste fill and directly under the back. This floor would be equipped with a track with which the fill below it had been placed. The track would now be used for the haulage of broken ore to the chute. The shoveler would work on the floor above the track (the floor below the machine) dropping the broken ore to the car on the track floor. (See Fig. 5 left.) The sorted waste would be thrown back, toward the raise. After the heading of the two floors is advanced about 30 ft., a track is placed on the top floor and waste drawn from the chute in the raise and trammed to the stope as filling, the track floor being partitioned off from the advancing waste fill for the purpose. Any waste sorted by the shoveler would be thrown back into this waste fill. This keeps the filling advancing with the heading and keeps the space between the fill and back to a minimum. This plan can be changed, mining one or three floors, according to local conditions. At the right-hand side of Fig. 5 is shown the method of drilling the face of the two floors with one set-up of the machine; in a wide stope, two machines could be used. At the left-hand side is shown the condition in the face immediately after blasting.

The first step is to place a cap 5 ft. from the last standing cap, using the broken muck as staging. In the meantime, the muck pile is lowered and as soon as possible a stringer is laid on the caps, upon which the vertical machine bar is placed (see Fig. 5). Drilling is started at once. When muck pile is lowered sufficiently another cap is placed, directly under the first cap. The muck pile is again used for the staging in placing this cap. The posts under the top cap are then placed. When the muck is removed on the lower floor, the lower posts are placed in position. Collar

braces are placed in position, sprags are placed on top of the timbers to the back, and the face is ready for blasting.

If the stope is over 16 ft. wide, two caps are placed end to end with the center from 6 to 18 in. higher than the extreme ends, depending on the width of stope, as shown in Fig. 5. These caps are carefully braced one to the other. On top of the highest point in this truss-set a post or sprag is carefully fitted to the ground, without blocking. This allows for all side pressure to be transferred to the timbers, holding the entire set tight. From two to four bridge sprags are placed on each cap, depending on the width of the stope and the conditions of the back. Two types of sprags are used; an upright made of 3-in. lagging 2½ ft. long or 5 ft. long, depending on the height of the back above the timbers. The upright lagging is placed on the cap and the space between the end of the upright lagging and the back is filled with 3-in. lagging 5 ft. long and placed parallel to the cap and fitted tight with wedges (as shown in Fig. 5). All lagging used as sprags is recovered and used as head-boards or flooring.

Fig. 5 also shows the method of crossboards through which the broken ore is passed to the car in the floor beneath.

Should the face contain a horse of waste, the machine is set up opposite the ore and the waste drilled with a minimum of holes to break it into as large pieces as possible for underground sorting.

This method of mining is especially adapted to the Hecla mine. It allows for continuous mining and continuous filling, keeping the space between back and fill to a minimum. It allows for the organization of the crew, according to the width of the stope, so that a regular schedule of operation may be maintained. The truss set, or toggle set, transfers the side weight to the back and vice versa. The fact that the waste fill is kept close to the face allows for the use of smaller timbers for caps and posts. The mine foreman estimates that 20 per cent. less timber is used by this method than formerly.

Timber Rilling

Timber rilling has been attempted in the Hecla and Intermediate orebodies, but, because of the weak walls, has been discontinued. The same type of set was used in the timber rills as in the horizontal stope, and the entire face of the rill was advanced three sets before the inclined floor was moved for filling. Drilling was started at the bottom of the rill, three sets being mined, and the machine moved to the floor above, where three more sets were removed. This was continued until the entire face of the rill was advanced three sets. Broken ore was allowed to stay in the rill, only enough being drawn to permit of room for a ladderway. When drilling was completed in the rill, the broken ore was drawn out, the inclined boards moved three floors, and the space under the boards

filled with waste. This method was varied in some cases, one or two sets being driven on each floor, instead of three. The timber rill has the advantage of lower cost per ton, but does not permit following stringers into the wall. It is not practical to use more than one machine in the entire rill at one time, because of the danger of falling objects. The operation is in cycles, making the supply of ore very irregular. The timber rill was found not to be adaptable to the Hecla or Intermediate orebodies. At the left-hand side of Fig. 4 is shown the method of starting a timbered rill stope. At the right-hand side is shown a typical condition after the stope is well advanced.

Untimbered Rills

The rill stope is an ordinary flat stope inclined to the angle of repose of broken muck.

It is necessary to start a rill stope adjacent to a raise from which waste filling can always be drawn as required. A rill stope may be started on each side of the raise. In beginning these stopes, the second, third, fourth, and fifth floors are started from the raise, and where the walls are poor (as at the right-hand side of Fig. 1), it is generally necessary to timber these floors with light timbers until the timbering is high enough at the raise for the waste to gravitate to the next ore chute, a distance of 50 ft. horizontally. In mining this first section, the muck is allowed to fall to the first floor above the level from which it is dropped through cross-boards directly into cars on the level below. The second floor should be mined about 30 ft. horizontally from the raise; the third floor about 25 ft.; the fourth about 20 ft.; and the fifth about 15 ft. After cleaning out this broken ore, poles 8 to 10 in. in diameter are laid on top of the first floor timbers to support the waste filling, leaving the drift and first floor open. The waste chute is then tapped on about the sixth floor and this space allowed to fill with waste until it comes to the angle of repose, as shown at the left-hand side of Fig. 2. In case this waste is coarse, care must be taken to prevent its rolling ahead at the bottom and necessitate shoveling the ore off the boards at the bottom. Keeping the angle of the rill sufficiently steep for the ore to be drawn off by gravity is a most important factor in the successful operation of the rill stope.

To prevent the ore mixing with the waste filling, 3-in. boards about 9 ft. long should be placed on top of the waste. It is advisable to put some kind of 3-in. scrap boards under the end joints of this floor to make it as smooth as possible.

The floors already started should now be advanced toward the ore chute, and additional floors begun. The second floor should be driven across the ore chute and the back gradually worked toward the angle of the waste fill. The ore chute should be approximately 50 ft. from the waste raise (as shown at the right-hand side of Fig. 2), and should consist

of a chute and manway. If the ore is not hard on chutes and the levels are not too far apart, necessitating too great an amount of ore to be handled, one chute might suffice. Furthermore, if the vein is wide enough, both the chute and the manway might be placed in one compartment. However, the three-compartment chute gives better satisfaction; the two outside compartments being used for chutes and the middle one for the manway. This second chute is used only to carry the ore while the first chute is being repaired, unless it is desired to sort the ore and waste at the bottom of the rill.

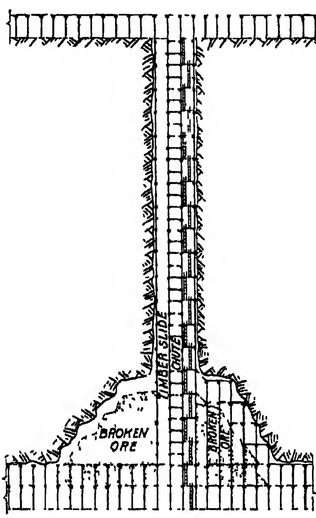


Fig. 1.

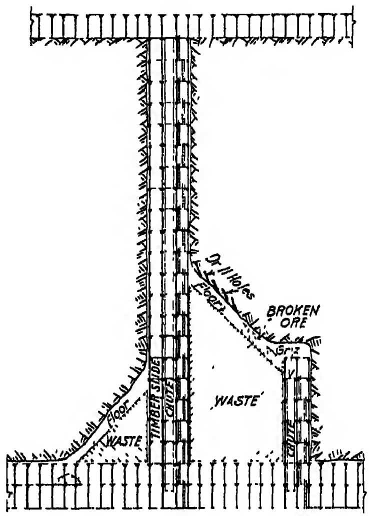


Fig. 2.

When the second floor has been driven across the ore chute and the other floors gradually worked to the angle of the waste fill, the ore should be cleaned through the crossboards, the poles for holding the waste continued on top of the first-floor timbers to the ore chute, and the timbers next to the ore chute lagged off to prevent the waste running into the chute. The 9-ft. boards laid to prevent the ore mixing with the waste should now be taken up and this space filled with waste to within 5 ft. of the back.

Up to this point, it will probably be necessary to stope in floors, or benches, and timber, as shown at the right-hand side of Fig. 1; but where the ground will permit, this work can be done after the shrinkage system, as shown at the left-hand side of Fig. 1. When this stage has been reached, there are several orders in which the work of mining, mucking and filling may be done, but the method found most satisfactory at the Hecla is as follows: When the stope is full of ore, the drilling is begun at the top of the rill adjacent to the raise, using air feed stopers. The holes are inclined toward the raise at nearly the angle of the waste fill (see

Fig. 2), so that when the next fill has been made, no difficulty will be found in loading the holes. It will probably be necessary to pull some of the ore out of the rill to make room to complete the drilling. This will enable the miner to operate his machine on a nearly level foundation, as shown in Fig. 2. This latter is an advantage because one of the greatest objections to the rill stope is that the miners must work from an incline. After the drilling has been completed to the bottom of the rill, the remainder of the ore is pulled into the chute and the slide boards taken up, half of them being piled in the timbers above the ore chute, and the other half pulled

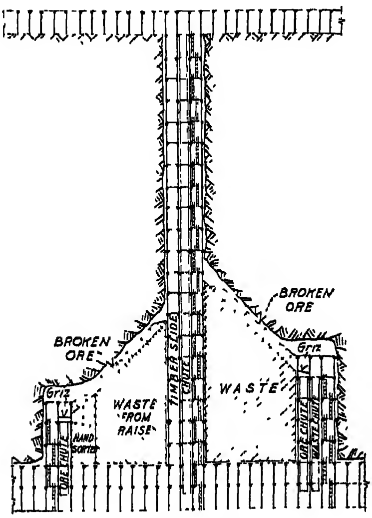


FIG. 3.

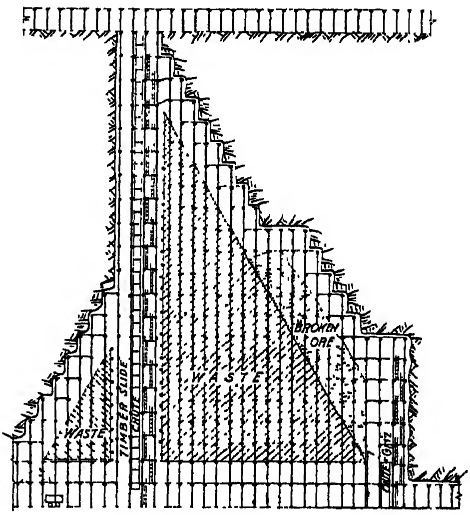


FIG. 4.

up into the raise where they are out of the way. The waste is again drawn in to fill the rill within 3 ft. of the back. The floor is again laid on top of the waste and the entire round blasted. Enough ore is pulled out to permit starting the drilling and the operations are repeated.

In ordinary ground, it is best to fill after each blasting. There will then be not over 8 or 10 ft. of the walls exposed at any one time, and the back will not be too high when the ore is withdrawn. However, in good ground, two or even three cuts may be made before filling, provided the walls are properly supported with stulls and the back is fairly firm.

As the operation of the rill is necessarily in cycles, the production is intermittent, and owing to the inclination only one or two machines can be safely operated on one stope. This makes it necessary to have enough rill stopes working at one time to secure the required production. If necessary, raises can be driven 100 ft. apart and two rills run into the same ore chute. If the necessary production can be secured without working the entire length of the orebody at one time, the cost of some of the raises can be saved by placing them 200 or 300 ft. apart. In the latter

case, when a rill stope is completed, the ore chute of the rill becomes the waste raise for the next stope, and the expense of the raise is saved. Only a preliminary sorting is attempted underground in the above systems, the attempt being made to take out only the larger pieces of waste. A careful sorting is made in the sorting plant, on the surface.

In the horizontal system (as used at present), all coarse waste is thrown back of the shoveler (toward the waste raise).

When desired, a small amount of waste may be sorted from the product of an untimbered rill by stulling off 5 to 10 ft. of room at the bottom

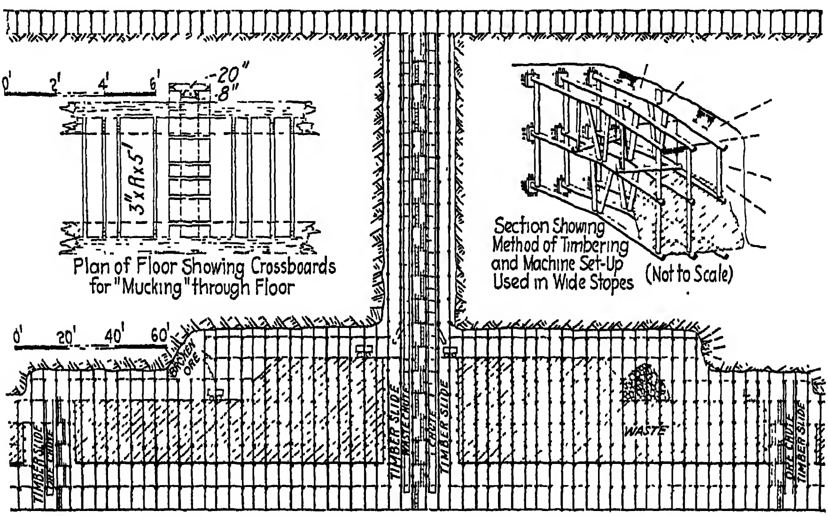


FIG. 5.

of the rill, next to the timbers of the ore chute. This will prevent the waste filling running to the bottom of the rill, and will leave room for a small amount of waste rock to be hand picked on the grizzly in front of the ore chute (see the left-hand side of Fig. 3) and thrown back into the space. This necessitates the building of a slide from these stulls to the top of the ore chute.

If more waste rock must be hand picked than could be handled in this way, a waste chute may be built in the compartment beyond the ore chute, as shown at the right-hand side of Fig. 3. With this arrangement the entire product of the rill may be run on the grizzly over the ore chute from which the waste rock may be picked off and thrown into the waste chute. Either of these methods may be used in the timbered rill.

Timbering

Drifts are timbered with stull sets, the caps being of random lengths and placed 5-ft. center to center (horizontally) with at least 1 ft. of heading

boards on each end of cap. The posts are 9½ ft. long. The first floor is timbered the same as the drift except that the posts are 8 ft. long. Floors are covered with fir lagging 3 in. thick, 5 ft. long of random widths. The bottom timber for the waste fill is laid on the cap of the first floor. This bottom is made of "corral poles," these being 6- and 8-in. stulls 10 and 15 ft. long. All rubbish (broken timber) available is placed on top of the corral poles to make the corral bottom tight.

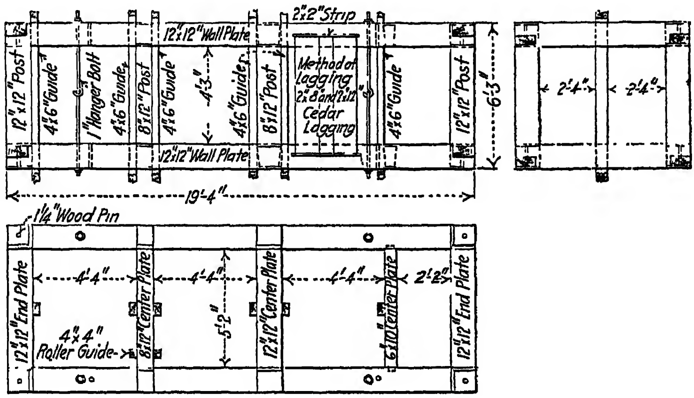


Fig. 6.—Shaft timbers, Hecla mine.

The simplest type of raise used is a large three-compartment (four set), wherein the posts are 8 ft. long, making the distance between floors 9 ft. The caps are placed 5 ft. center to center, measured horizontally, and in a line directly over the caps beneath. Working raises are broken the width of the ledge, or a minimum of 10 ft. and a maximum of 16 ft., the specifications calling for at least 1 ft. of heading boards on each end of cap. The

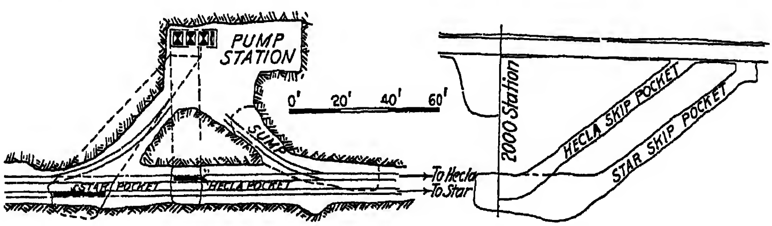


Fig. 7.—2000-ft. station, Hecla mine.

timber slide and manway are placed in the other compartments, the chute being in the center. The chute is constructed of 6-in. by 10- in. by 9-ft. timbers and the timber slide of 3 in. by 12 in. by 9 ft. This type of raise is shown in Fig. 2.

Should the raise be located where excessive side weight would be expected, the chute set is reenforced with two extra caps and the distance between floors increased to 10 ft. In this case, the chute is made of

3-in. by 5-ft. lagging of various widths, doubled. The slide is made of 3 in. by 12 in. by 10 ft. as shown on Fig. 1.

Our latest type of working raise calls for five compartments, having two slides, two chutes and one manway. The timber slides are placed in the extreme ends of the raise and the manway in the center compartment. The other two compartments are used as chutes. The caps are placed 5 ft. center to center measured horizontally and the four center caps are 5 ft. center to center vertically. The two end caps are 10 ft. vertically. The chute is made of 3-in. by 5-ft. lagging of various widths, doubled.

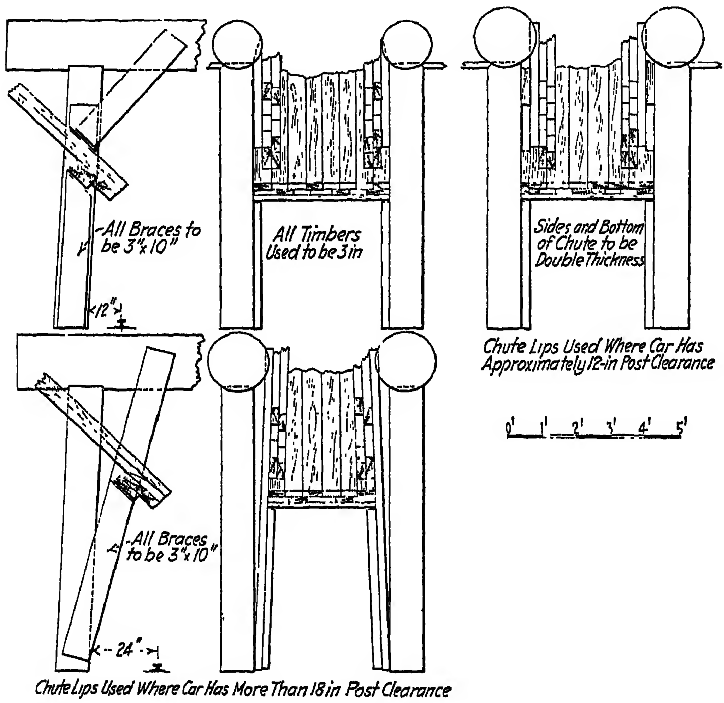


FIG. 8.—CHUTE LIP, HECLA MINE.

The slide is made of 3-in. by 12-in. by 10-ft. timber. This raise allows the repairing of one chute without interference with the passage of ore or waste through the raise. It also allows the hoisting, or lowering, of material to workings on either side of the raise (see Fig. 5).

Raises of the shrinkage type are driven, when it is planned later to remove the timbers from the raise, making a rock chute. This type of raise is driven 8 ft. wide and 15 ft. long. It is timbered by dividing the raise into three compartments, by placing two caps every 5 ft. vertically. The side of these caps nearest the end of the raise is double lagged, with 3-in. material and the two end compartments are used as chutes, the rock wall of the raise forming three sides of the chute. The

center compartment is used as a timber slide and manway. When it is desired to remove the timbers from this type of raise, the broken ore or waste is drawn from the two chutes and the raise emptied. Work is started at the bottom of the raise removing the timbers, which are hoisted up the raise to the level above. As timbers are removed, ore or waste is dropped in from the top, giving the men a substantial foundation upon which to work. The bottom of this type of raise is started with substantial timbers, 18-in. caps and 12-in. posts being used. When the timbers are drawn from the raise, 10-in. poles are placed on top of the timbers.

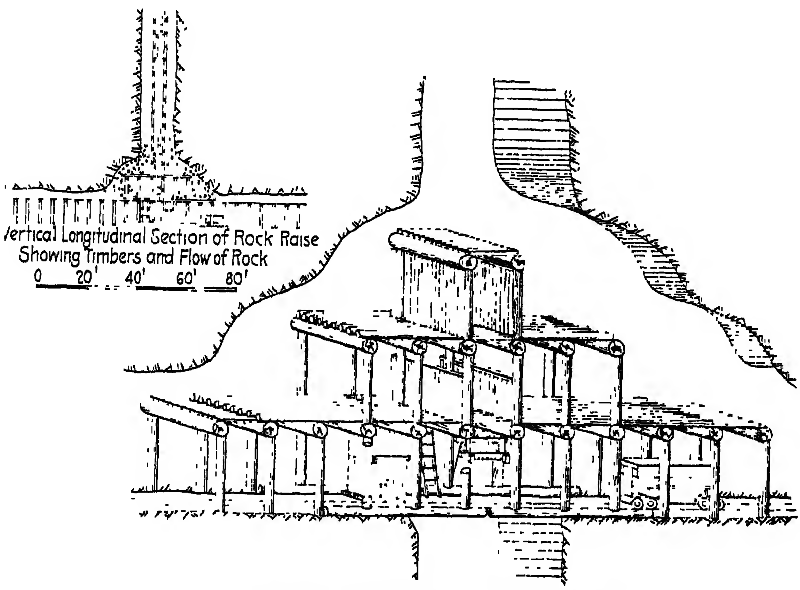


FIG. 9.—SHRINKAGE RAISE, HECLA MINE.

This type of raise can be used in connection with a shrinkage stope but is not recommended for any stope wherein waste fill is necessary; it is shown in Fig. 9. When lacing or timbering off the waste fill from a raise, new caps are placed 5 ft. apart vertically and about 2 ft. from the center of the outer raise cap; 3-in. lagging 5 ft. long is nailed to the inner side of these caps. Some old lagging can be used for this purpose. This method allows the retimbering or repairing of a raise without endangering the waste fill.

Timber is delivered to the place of work in the stopes by a crew consisting of three men. No preservatives are used on the timbers. Timbers are cut to proper length on the surface. The cutting of "daps," or other framing, is done underground, where the timber is to be used. Posts are cut on the surface and, being standard, can in most cases be used without further framing. Stringers are ordered special and are

framed on the surface, although a few standard stringers 16 ft. long are stored on each level for emergencies. Some lagging is recovered, to be used again, but the round timber is left in place, the waste filling being placed around it. Old timber recovered from repairing is sent to the surface and used as fuel in the heating plant. Whenever possible, standardization is attempted. Fig. 8 shows a standard chute lip. This construction requires the minimum amount of timber, for the replacing of timbers destroyed by wear, and gives assurance of a safe clearance for the mine cars when under the chute. Fig. 9 shows a standard "draw chute" used for drawing broken ore or waste from one level to another.

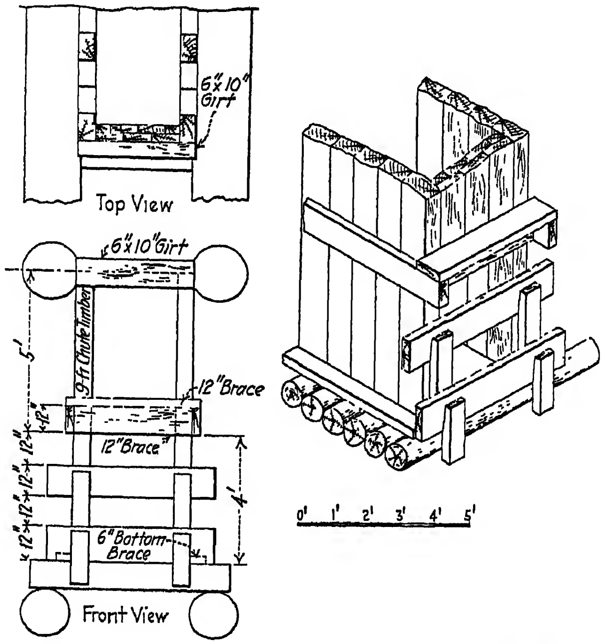


FIG. 10.—DETAILS OF DRAW CHUTE, HECLA MINE.

Underground Sampling and Check Sampling of Product on Surface

Milling ore is sampled at the head of the mill by an automatic sampler (Vezin). Waste is not sampled, but an effort is made to watch the waste product very carefully. Some years ago an entire railroad car of waste was put through the sampler, giving a result of silver 0.7 oz., lead 0.41 per cent. In calculating the metallic content of waste, the above assay was used. No attempt is made to sample the crude at the mine, the smelter sampling being used in all cases. Concentrates and slimes at the mill are sampled as loaded, a shovelful being taken from each wheelbarrow or ore buggy.

Tonnage is distributed to each level in proportion to skips hoisted from that level. A record is kept of cars from each chute or section dumped into skip pockets on each level; this gives a figure for each chute or section, which we call tonnage, but which is really volume. All cost per ton data for the different sections are obtained from these figures.

For 3 months a sample was taken at the lip of each chute in the entire mine (trainmen would take a shovelful from each car). This sample was cut up weekly (by coning and quartering) and assayed. Its assay was spread over the tonnage from the chute for the same period. The total metallic units of silver and lead for the 3 months was thus computed and compared with the metallic units hoisted. This latter figure was obtained by smelter assays on crude ore, assay on mill heads, and the figures obtained for waste. It was found that during the 3 months the samples taken in the mine would run higher than the actual metallic content by the following percentages: silver, 12.401 per cent.; lead, 10.761 per cent.

Certain sections, where the assay is low or variable, are sampled every carload and the sample cut up and assayed every week. Results are given to the foreman immediately. Grab samples are taken by the mine foreman at different times for his own information. All samples (except grab) are coned and quartered in the mine, the larger pieces being broken with a hammer. About 100 lb. of the sample is brought to the surface, where it is crushed, coned, and quartered to about 2 pounds.

Loading Machines and Scrapers

The Hecla mine has three Armstrong No. 11 Shuveloders in use. Two are in the Star crosscut (one extra in case of breakdown) and one in the Hecla 2000-ft. east drift.

Tramming and Haulage

The five haulage motors are class A-7; 24-in. gage; 7000-lb. chassis; Baldwin-Westinghouse, bar-steel, storage-battery locomotives equipped with two only V-50 by 4 60-volt, 1150-r.p.m. motors, steel-tired wheels; 66 cells Edison type A-8-H battery; battery crate in one section with rollers to facilitate removal. Motors are outside hung, the frame being outside of the wheels, which are 24 in. in diameter. Journals are 3¼ in. in diameter and 4½ in. long. Type 161A controller, motors are straight series connected. Two-step gear reduction, motor pinion to jack-shaft or intermediate shaft gear 15 to 66, or 1 to 4.4; jack-shaft to main shaft or axle shaft (pinion to gear), 16 to 76 or 1 to 4.75. Total weight on drivers (all wheels) is 9500 lb. Drawbar pull, accelerating, is 3000 lb.; running, 2000 pounds.

Data from the 1600-ft. level may be taken as an average: Bottom dump cars with two four-wheel trucks are used. They weigh 4040 lb.

each and hold 5.2 tons muck. A train consists of eight cars; the average number of trips in an 8-hr. shift is seven, but ten can be made. The average grade against the empty train inbound is 0.75 per cent.; the maximum 1.44 per cent. Speed of the empties against grade is 4 mi. per hr.; loads with grade 6 mi. per hr. A standard section rail is used, weighing 35 lb. per yard.

Underground Storage and Dumping

Where possible, an effort is made to allow for the storage of broken ore underground and as far as possible the chutes are kept full. The skip pockets have a capacity of from 70 to 500 tons. A plan of the 2000-ft. station, the latest cut at the Hecla mine, is shown in Fig. 7. The Hecla skip pocket shown thereon has a capacity of 500 tons. The Star pocket has a capacity of 1040 tons. The broken ore is drawn direct from the skip pockets to the skip, no measuring pockets being used.

Hoisting

Main Hoist.—The main hoist is of the Ilgner type with Ward-Leonard control. The two reels (with a center diameter of 5 ft.) are each equipped with 2500 ft. 3/8- by 5½-in. flat sewn rope. Lane clutches and post brakes are used. The reels are mounted on a common shaft with a 375-hp., 500-volt, 60-r.p.m., d.c., shunt-wound motor. Excitation is furnished by a 25-kw., 125-volt generator mounted on a motor-generator shaft.

The motor-generator set is self-contained, having a cast-iron subbase, four bearings, and a shaft. The motor, or driving element, consists of a 450-hp., three-phase, 60-cycle type HF induction motor, with wound secondary, designed to operate between 2080 and 2300 volts. The generator is 450-kw. 720 to 580 r.p.m., with commutation poles to handle full-load current from 0 to 600 volts. On the shaft between the motor and the generator, a 29,000-lb. cast-iron spidered, laminated, steel-rimmed flywheel is mounted. On the generator end of shaft is mounted the 25-kw., 125-volt exciter which serves both the generator and the d.c. hoist motor. A slip regulator is provided, operating through the secondary of the a.c. motor, to reduce automatically the current to the induction motor, allowing the flywheel to give up its stored energy. This eliminates the intermittent heavy current demand on the WWP's line, occasioned by a sudden heavy load on the generator.

Control is centralized in the hoisting engineman's platform. It consists of a control lever, centrally located, governing speed and direction; one brake lever and one clutch lever for each reel. Safety is provided by Welch automatic governors, one connected to each reel, which control overwind, underwind, and overspeed. All electrical equipment is of Westinghouse manufacture.

The following data are from actual operation, not from computation:

The skips weigh 6422 lb. each; the cages, 4584 lb. each; and the water-bailer 4168 lb. each. Directly under the head-house floor, and adjacent to each compartment of the shaft, pits are provided, each holding one skip and one cage, making interchange between skips and cages but a matter of moments, though only one skip or one cage is hanging on the rope at any given time. The water bailers as well as spare skips and cages are located in handy storage, ready for any emergency.

One rope, 2500 ft. long, weighs 8990 lb.; a skip holds 2.8 tons muck; cage accommodates 18 men on two decks; bailer holds 1150 gal. Diameter of hoist reel at various levels as follows:

With skip in dump	11 ft. 2 in.
With skip at 900-ft. level	8 ft. 9 in.
With skip at 1200-ft. level	8 ft. 0½ in.
With skip at 1600-ft. level	6 ft. 11½ in.
With skip at 2000-ft. level	5 ft. 3 in.

Hoisting speeds, skip with muck balanced are:

From 900-ft. level, 2241 ft. per min	1.03 min. per skip
1200-ft. level, 2161 ft. per min	1.42 min. per skip
1600-ft. level, 2039 ft. per min	1.44 min. per skip
2000-ft. level, 1845 ft. per min	1.74 min. per skip

The maximum hoisting speed is 2364 ft. per min.; the average is 1.4075 min. per skip. Normally the hoisting is balanced, but we can hoist unbalanced.

Jitney Hoist.—The jitney hoist is a General Electric type 1, Form M, 75-hp., 450-r.p.m., 2080- to 2300-volt, 60-cycle, three-phase, a.c. induction motor, with wound secondary; through single reduction gear, 26 to 149 teeth or 1 to 5.73, to hoist drum running at 78.5 r.p.m. Rheostatic control is had through the secondary winding of the motor. A round rope, ⅞ in. in diameter, 2500 ft. long, 6 by 19 Blue center steel is used; it weighs 2622 lb. and runs at 730 ft. per min. The cage weighs 2400 lb. All running is done unbalanced. This equipment is for relief service only, no muck, timbers, or steel.

Pumping

On the 2000-ft. station, there are two 500-gal. per min., type MT, four-stage, Cameron turbine pumps, each direct connected to a 200-hp., 1750-r.p.m., 2200-volt, 44-amp., three-phase, 60-cycle, Westinghouse Type CS induction motor a.c.

On the 1200-ft. station, there are two 425-gal. per min. Aldrich quintuplex plunger pumps, each connected through single gear reduction to a 175-hp., 435-r.p.m., 2300-volt, 41-amp., three-phase, 60-cycle Westinghouse Type CCL induction motor a.c. and one 650-gal. per min. type MT, four-stage, two units in series, Cameron turbine pump with one 300-hp., 2200-volt, 70.5-amp., three-phase, 60-cycle, Westinghouse

Type CW induction motor, a.c., between the units on a common shaft. Rheostatic control is through the wound secondary of the motor.

Air Compression

All compressors are Ingersoll-Rand imperial type 10. There is one cross-compound, two-stage; low-pressure, or first-stage, cylinder 17 in. in diameter; high, or second stage, 10 in. in diameter; with 14-in. stroke, running at 184 r.p.m.; belted with a 100-hp., 705-r.p.m., 2200-volt, 24.9-amp., three-phase, 60-cycle Westinghouse Type CS Induction Motor, a.c. Displacement is 720 cu. ft. per min.

One cross-compound, two-stage; low-pressure, or first-stage, cylinder is 23 in. in diameter; high, or second stage, 13 in. with 16-in. stroke, running at 150 r.p.m., belted with a 150-hp., 600-r.p.m., 2080-volt, 40-amp., three-phase, 60-cycle, General Electric type 1, Form K, induction motor, a.c. Displacement is 1100 cu. ft. per min.

One cross-compound, two-stage; low-pressure, or first-stage, cylinder is 28 in. in diameter; high or second-stage cylinder, 17 in.; with 24-in. stroke, running at 119 r.p.m., bolted with a 300-hp., 514-r.p.m., 2080-volt, 80-amp., three-phase, 60-cycle, General Electric Type I, Form M, induction motor, a.c., with wound secondary, with rheostatic control through secondary. Displacement is 2200 cu. ft. per min.

The displacement given is in cubic feet of free air at normal atmospheric pressure.

Ventilation

The Hecla mine is ventilated by natural methods, such ventilation being materially helped by a raise 1500 ft. long extended from No. 3 tunnel (approximately the elevation of the collar of the shaft) to the surface. The general direction of air currents is controlled on different levels by doors. Artificial ventilation is used only in blind drifts.

The 2000-ft. Star crosscut is equipped with a No. 8 Buffalo blower used as an exhauster directly connected with a 10-hp. motor, running at 1740 r.p.m. This exhauster is placed at the mouth of the crosscut; galvanized-iron 22⅝-in. ventilating pipe is used in this crosscut. At a point in the crosscut, from 300 to 600 ft. from the face, a No. 5 blower is used to blow air through a 10-in. galvanized iron pipe to the face. This blower is operated by a 5-hp. motor.

When a raise, blind drift, or crosscut is being driven, and extra ventilation is needed, a No. 5 Buffalo blower is installed until such work is completed.

Lighting

All lighting, with the exception of locomotive headlights and hoist-room emergency running lights, is 120 a.c. Mazda lamps from 40 to 100 watts are used for general illumination in surface buildings. Yard and outside lights are 300 watt in cone reflectors, from 15 to 85 ft. above

ground, depending on conditions and service required. The hoisting-rope areaway is lighted from reels to sheaves by 40-watt lamps. In addition to the a.c. lighting in the hoist room, direct current from the 25-kw. exciter of the motor-generator set is connected to one side of a double-throw, double-pole switch (a.c. on the other, lights in the center) to insure running lights for the hoistman even if the a.c. circuit is interrupted. These lamps remain lighted as long as sufficient energy to move the hoist is being generated.

The shaft is lighted directly opposite each skip pocket to enable the cager to see into skips while loading. These lamps, two 40-watt, with their terminals incased in sealing compound, are inset under the wall plates to prevent breakage from falling rocks, etc.

The stations, under 10-ft. back, use 40-watt lamps; over 10-ft., 75-watt. The number is governed by the size of the station and the amount of travel or work.

The drifts are served by No. 6 B. & S. solid wire, 120 volt with booster-transformer service, and 40-watt lamps spaced from 30 to 85 ft. apart depending on curves and other conditions. Red 10-watt lamps are placed in front of powder magazines, capping benches, etc. Exit signs, showing direction of surface manways, are lighted by 10-watt green lamps.

There are no electric lights in the stopes above the levels. Each man uses a carbide lamp generating acetylene gas. The type generally used burns approximately 4 hr. on one filling of calcium carbide.

Telephone

The underground telephone system is bridging, non-selective, and universal ringing. Talking circuit is fed by dry cells; the ringing circuit is 120-volt a.c. Phones are installed in the main and jitney hoist rooms, top station, fire-tunnel station, and all shaft stations. Interchange in the main hoist room connects the mine line with the surface lines. Magneto ringing is furnished for emergency use. Underground and top-station phones are Stromberg-Carlson Mine-a-phones in steel waterproof cases; 16 c.p. carbon globes inside the case counteract dampness. The calls are in accord with the hoisting signals.

The shaft-signal system consists of a three-wire return-call audible and visible signals, mounted in and on a galvanized-iron box, wherein a Benjamin No. 8299A industrial type buzzer, taking 0.17 amp. at 120 volts and a 20-watt lamp are connected in multiple. A watertight push button is mounted on the lower front of the box, in series with the line and with the buzzer and light for signaling and answering. The train crews and certain mechanics in the drifts are called to the station by alternately opening and closing the switch controlling the lights.

In the stopes, a buzzer and light are mounted in the drift near each tugger or other hoist and controlled by a push button in the manway or slide to raise or lower steel, timbers, etc.

The proposed semaphore system for the long Star tunnel will be manually operated from either end or intermediately by suitable switches; red lamps for danger and green for clear track. With neither red nor green showing, caution should be used, as this would indicate some derangement of the system.

Records of Unit Production

The total tonnage of ore hoisted from the Hecla mine during the calendar year 1921 was:

	Short Tons
Horizontal stopes	190,054.86
Timbered rill	33,686
Development on ore	28,710.46
Repairs	2,136
From east orebody	13,238
	267,825.32

Note.—Ore from east orebody came from large storage raise. Broken ore from shrinkage stope, development on ore, and repairs was placed in this raise. Exact tonnage from each source not known.

All stoping is done by day labor. A few contracts have been let in raises, drifts, and crosscuts; in this case contract is based on advance in feet.

Tons per man per hour for miners in stopes.

Horizontal stopes	0.712 ton per hour
Timber rill	0.800 ton per hour
Shrinkage stopes	not available

On development in ore, 1.0138 tons per hour are produced by men.

No record is kept of the tonnage from crosscuts in waste, with the exception of the Star crosscut. The following data cover the operation of the crosscut for the months of March, April, May, and June, 1922.

Period advance	1,369¼
Period tonnage	11,390.3

	Shifts	Hours	Cost
Miners	893⅜	7,174	$4,250.04
Shovelers and mechanical mucker (includes pusher)	663⅝	5,309	3,337.14
Motor	277⅞	2,223	1,319.92
Timbering	1¼	10	5.95
Ditchman	95	760	380.00
Shift boss	143	1,144	822.28
Track and pipe	118⅜	947	562.25
	2,192½	17,540	$10,677.58

Tons waste per hour	0.649

From all underground miners 0.749 ton per hour is produced; this includes shovelers, trammers and timbermen on ore, both stoping and development, but does not include miners or tonnage on waste.

For all underground labor on ore 0.4428 ton per hour is produced; waste is not included.

Production per man per hour for surface labor (all working force on surface exclusive of office) is 0.965 ton per hour. Includes all surface and surface supervision (not sorting plant) and is calculated against ore tonnage only. It includes surface construction on surface.

Production per man per hour for total organization including office force is 0.295 ton per hour. This includes the total organization exclusive of mill and sorting plant and is calculated against ore tonnage only.

Classification of labor, expressed in percentage of total.

	Shifts	Hours	Per Cent.
Mine (ore only)	74,861⅛	598,889	65.92
Mine supervision	730	5,840	0.65
Surface	32,130¾	257,046	28.29
Surface supervision	2,555	20,440	2.25
Office	3,285	26,280	2.89
	113,561⅞	908,495	100.00

The labor turnover is as follows:

1921	Average Daily Shifts	Days Worked	Men Hired	Average per Day	Men Quit	Average per Day
Jan.	373.36	26½	102	3.85	89	3.36
Feb.	361.34	24¾	61	2.46	95	3.84
March	328.65	28	27	0.96	71	2.54
April	309.51	26⅜	121	4.59	96	3.64
May	329.30	26⅛	111	4.22	101	3.84
June	400.93	26	148	5.69	89	3.42
July	395.15	26¼	116	4.42	105	4.00
Aug.	413.36	28	127	4.54	95	3.39
Sept.	446.98	26¼	74	2.82	82	3.12
Oct.	471.23	26⅛	99	3.76	67	2.54
Nov.	472.30	26⅛	57	2.16	60	2.28
Dec.	480.55	27¼	55	2.02	62	2.28
	398.837	26½	91.5	3.45	84.3	3.18

Labor turnover 0.797 per cent. per day.

Records of Units of Supplies

Per ton of ore produced, required the following amounts of explosives:

Horizontal stopes	0.361 lb. per ton
Timber rill	0.322 lb. per ton
Development on ore	1.068 lb. per ton
Shrinkage stopes	not available

For stulls, 1351 lin. ft. of timber per 1000 tons were used; for lagging 12,364 board feet per 1000 tons were required.

Based on the 1921 output, from the Hecla there were hoisted	267,825 tons
From Star skip pocket	16,085 tons
Total	283,920 tons

	Total Kilowatt-Hours	Total Horsepower Hours	Per Ton Kilowatt-Hours	Per Ton Horsepower Hours
Mining (compressed air)	1,323,000	1,772,820	4.660	6.244
Haulage and hoisting	1,136,420	1,522,803	4.002	5.363
Pumping	1,662,990	2,228,407	5.857	7.848
Ventilation	64,800	86,832	0.228	0.306
Lighting, underground	139,550	186,997	0.492	0.659
All others: sorting, mechanical, sawmill, etc	473,540	634,543	1.668	2.235
Total consumption	4,800,300	6,432,402	16.907	22.655

Lighting	0.005
Hoisting	0.025
Pumping	0.036
Mechanical department	0 001
Compressor	0.029
Ventilation	0.000
	0.096

To produce 1 ton, 3588.9 cu. ft. of free air are displaced.

The supplies used, but not taken into account above, are 10.47 per cent. of the total cost.

Ore Recovery

The total ore recovery after sorting, milling, and smelting are 82.767 per cent. silver and 83.173 per cent. lead.

Disposition of Ore after Reaching Surface

Ore is dumped into the storage bin on surface from which it is run through the sorting plant. Crude ore, when sorted, goes to the smelter. Waste, when sorted, is returned to the mine or shipped out over the railway for railway use. Milling ore is shipped daily to Gem, Ida. Mill concentrates, slimes, and flotation slimes are shipped to B. H. & S. Smelter at Bradley, Ida. Tailings are washed into the creek.

Safety and Welfare Work

No special welfare work is attempted. Married men are encouraged to own their own homes and many receive assistance from the company in financing their homes.

Mining Methods of the Morning Mines

By Frederick Burbridge, Wallace, Ida.

(New York Meeting, February, 1923)

The Morning silver-lead-zinc mine of the Federal Mining & Smelting Co. is about two miles northwest of Mullan, Ida.

The lode is a metasomatic fissure vein. The orebody is approximately 2000 ft. long within the bounds of the property and varies in width from knife edge stringers to 50 ft. It has been developed to a depth of 3600 ft. and stands nearly vertical in Revett quartzite, which strikes north 65° west.

The high-grade veins and lenses are generally mixed with waste (siderite, quartz, and quartzite). The average grade of ore to sorting plant in 1921 was 4 oz. silver per ton, 8.6 per cent. lead, and 4.1 per cent. zinc. Wall rock stands only a short time without timbering. The minimum angle of gravity flow is 45°. The ratio of waste sorted out to ore is 6 per cent. No gases are generated underground and temperature has no influence on mining operations. Fresh air is forced down the air raise and up the main shaft.

Drifting and Stoping

Main drifts 12 ft. wide and 13 ft. high are driven every 200 ft. (vertically) and are connected by raises 125 ft. apart. These raises consist of three compartments (chute, tramway, and timber slide) and are for the purpose of ventilation, access, and hoisting supplies to stopes. Ground is stoped between raises.

Chutes are placed every 25 ft. for drawing off ore broken in stopes.

Waste for filling is obtained from waste raises off the stope in walls or waste is dumped from the level above down the chute compartment of raises, which is bulkheaded so that waste can be drawn off at any floor as required.

Timbering

Posts and caps for square sets (varying in diameter from 6 to 24 in.) are cut to length in a sawmill on the surface. No preservatives are used. Timber is taken on a flat car along drifts to a raise nearest the working place and hoisted as required. Practically no timber is salvaged for

reuse except for building bulkheads, when conveniently available, and for waste corrals.

Drilling and Blasting

The drills used are: Ingersoll-Rand dry stoper (B.C. 21 hammer drill), Ingersoll-Rand No. 248 Leyner-Ingersoll, Denver Rock Drill No. 55 Clipper, and Denver Rock Drill No. 60 Waugh dreadnaught for sinking.

Experiments are being made with the following drill steel: 1¼-in. round Leyner steel (International High Speed Steel Co.); 1-in. Bulldog, quarter octagon, solid (International High-Speed Steel Co.); 1⅛-in. cruciform (International High-Speed Steel Co.); and 1-in. vanadium steel, quarter octagon (Red Star) (Colonial Steel Co.). Single-taper square bits with centers higher than edges are used.

The arrangement of holes is as follows: In stopes, all uppers lean toward the breaking face, and in drifts the position depends on size of drift and hardness of rock. Drift holes are tamped and all holes are fired by fuse except in shaft sinking, where they are fired electrically. Large boulders are drilled with jackhammers. du Pont 30 and 40 per cent. gelatin dynamite, 1⅛-in. sticks, are used.

Loading, Tramming, Etc.

An air-driven Hoar mucking machine is used in a drift on one level.

For haulage on levels 4-ton trolley motors are used; a 4½-ton storage-battery motor is used on two levels. In the main tunnel, a 6-ton trolley motor is used. In the main haulage 60-cu. ft. bottom-dump cars are used; in drifts, 27-cu. ft. side-dump cars; in sublevels in stopes, 17-cu. ft. end-dump cars. Rails are 25 lb. on levels and 45 lb. on the main tunnel. The track is 24-in. gage and has a 0.5 per cent. grade.

Cars are dumped into a skip pocket on each level and drawn off, as required, to skips that dump into a bin or underground pocket at top of shaft, where it is drawn off for a train loaded at the top station in the main tunnel, approximately 2 miles from the mill storage bin.

The main hoist is a direct-acting, double-reel, Curtiss hoisting engine, made by Fraser & Chalmers. Its nominal rating is 600 hp. Cylinders are 20 in. in diameter with 60 in. stroke. They are direct-connected to a pair of reels for winding 2600 ft. of 6 by ½-in. flat rope on each. The reels are fitted with steam-operated clutches and post brakes and the engines have steam reverse.

The hoist is operated by compressed air at 80 to 90-lb. pressure. The air is supplied to the hoist at a temperature, such as is delivered by a two-stage 2500-cu. ft. compressor placed adjacent to the hoist, in conjunction with the main air plant, which has a maximum free-air capacity of 10,800 cu. ft. per min., of which 5200 cu. ft. is supplied by direct-

connected water power. The maximum capacity is reached only during high-water periods.

The auxiliary hoist used for hoisting men, timber, and supplies, consists of a geared 300-hp., electrically driven, single-reel engine. Cages have a capacity of 18 men and the skips 3½ tons. The speed of the main hoist is 1600 ft. per min. in balance, and the electric hoist, 800 ft. per min.

Pumping

At the 1800-ft. station is a triplex Gould pump lifting 1050 ft. It has 6 by 12-in. plungers and a capacity of 1750 gal. per min. At the 2200-ft. station is a triplex Gould pump lifting 400 ft. It has a capacity of 200 gal. per min In addition, on each pumping station is a No. 9 Cameron air pump as an auxiliary. It pumps from 2200 to 1800 ft., thence to 1200 ft., and finally relayed to the 800-ft. level. The average water pumped is approximately 80 gal. per min.

Ventilation

The intake is through No. 6 tunnel, near the end of which is a Sirocco fan of 60,000-cu. ft. capacity driven by a 100-hp. motor, which forces air down the air raise at the east end of the mine. There are doors on all levels, except the lowest, to prevent short-circuiting of the air. There is an exhaust fan of 40,000-cu. ft. capacity near the collar of the shaft, and dead air is naturally drawn up the old shaft to No. 5 tunnel level, thence to old raises connecting with higher levels to the surface.

Lighting, Signals, and Communication

Electric lights are installed in the main tunnel, stations, and drifts. Carbide lamps are used by workmen. Hoist signals are by electric bells and flashlight. A block system of danger signals is used in the main haulage tunnel to prevent collision at curves and at the portal. Telephones on each station level connect with other levels and hoist rooms, head house, and office; connections are established by signals.

Unit Production

Day labor is employed except for contracts on development and driving drifts and raises, based on a cubic foot and linear foot unit. The production is 3.47 tons per man-hour for all miners and 0.288 man-hour per ton for all miners. These men include all machinemen on ore. If all men in stopes and on development work be included as miners the figures would be 0.64 ton per man-hour and 1.56 man-hours per ton. The production is 0.247 ton per man-hour for all labor (including office) and 4.049 man-hours per ton for all labor. Labor is classified as follows:

	Per Cent.
Superintendence, foremen, shift bosses	3.6
Machinemen	10.2
Timbermen and carpenters	28.6
Muckers	12.3
Trainmen	8.5
Shaft repairmen	5.1
Cages, powdermen, nippers	2.0
Trackmen and pipemen	1.8
Blacksmith, machinists and electricians	6.3
Hoistmen, oilers, and helpers	4.7
Surfacemen	5.3
All development labor	11.6

The total labor cost is 67 per cent. of the total cost of mining.

Units of Supplies Used

The explosives used were 1.69 lb. per ton.

The timber used was 1.12 board ft. per ton (12 by 12-in. 10 by 14-in. shaft timber, guides, etc.); stulls, 1.23 lin. ft. per ton; lagging 11.18 board ft. per ton.

The total power, including development and timber hoists, as segregated as follows, is 1.5 hp. per ton:

	Horsepower
Mining (ore breaking)	0.48
Haulage and hoisting	0.39
Pumping	0.06
Ventilation	0.15
Lighting	0.02

Other supplies formed 8.27 per cent. of the total cost of mining. The total cost of all supplies was 33.0 per cent. of the total cost of mining.

Disposition of Ore after Reaching Surface

After reaching the surface, the ore is sorted, part is shipped direct, the waste goes to the dump, and the balance goes to the mill.

Safety and Welfare Work

Safety bulletins are posted and the usual precautions are taken to make everything as safe as possible. Bonuses are given to bosses having the best records—that is, the fewest accidents.

The welfare work includes free insurance, club house, hotel, and literature.

Filled Stopes

A FILLED stope is one in which the support for walls and men and, at times, for the back of ore, is furnished by waste rock or sand tailings. The filling may be rock sorted out in the stope or from the walls, development, or the surface. The orebody is mined in sections, each practically an open or timbered stope, filled wholly or in part before adjacent ground is attacked. As a rule, the filled-stope method is limited to overhand flat-back, stepped-face, or rill stopes. The examples given are Copper Range, Iron Cap, Verde district, Zaruma district, Lucky Tiger, and Silver King Coalition.

Mining Methods of the Copper Range Co.

By W. H. Schacht,* E. M., Painesdale, Mich.

(New York Meeting, February, 1923)

The operations of the Copper Range Co. are located in the Michigan copper district at the southern end of the Keweenaw Peninsula, 8 miles southwest of Houghton (the center of the district). All the producing mines are situated in a long narrow zone paralleling the peninsula and occupying the central portion. This zone is about 25 miles long and 3 to 4 miles wide. The important mines, in their order from northeast to southwest, are, Mohawk, Ahmeek, Allouez, North Kearsarge, Wolverine, South Kearsarge, Centennial, Calumet & Hecla, Osceola, Franklin, Quincy, Isle Royale, Baltic, Trimountain, and Champion; the last three are operated by the Copper Range Co.

The existence of copper deposits in this district became known soon after the discovery of America. Many references by early explorers show that copper was found in the possession of many of the American Indians that inhabited the Great Lakes region. There is abundant evidence that exploration of these copper deposits extended into prehistoric times; and some indication that the American Indians carried on operations shortly before the advent of white men. An account of the finding of primitive mining appears in the Government publications of 1850–51 by Foster and Whitney.

Although earlier expeditions by white men had been made to the Keweenaw peninsula, an attempt to commercialize the district was not made until after 1765, when Alex. Henry, a British subject, searched the shores of Lake Superior for mineral treasure. His first expedition took him to the Ontonagon River, where he found the famous mass of copper now on exhibition at Washington. In 1771, he started operations along this river with a crew financed by English capital; but after much difficulty and only meager success he found only detached masses while in search for a major deposit.

There is no record of any exploration from 1774 to 1820. In 1838, Dr. Douglas Houghton was appointed state geologist. Prior to that time he had made several visits to this region; and was the first white man to find

* General Manager, Copper Range Co.

copper in place in the bedded rocks. It was after the publication of his reports, in 1843 and later, in which he discussed the general prospects of profitable mining and gave to the world the true condition of the Keweenaw peninsula, that the first real mining was started. This, however, was not done in the conglomerate and amygdaloid beds from which the product of today is wrought, but in true fissure veins that crossed the conglomerate and interbedded amygdaloidal trap sheets. Copper, with small amounts of silver, occurred in the native state in these fissures in masses up to many tons.

During the 40's, mining operations were carried on at both ends of the peninsula: the Cliff mine at the northern, at Eagle Harbor, the Minnesota (now the Michigan) at the southern, and the Pewabic at the central portion near Portage Lake. It was not until 1856 that the Quincy amygdaloid and the main Pewabic beds, which are the oldest beds worked today, were found. Other mines were opened thereafter in rapid succession, and the first stamp mill was built in the late 50's. The Calumet conglomerate was discovered in 1864 and opened in 1869. This has been the premier producer of the district: 65 per cent. of the district's production came from it in 1885, when the district produced 43 per cent. of the total production of the United States.

Geology

The rocks in which the copper ore of this district are found belong to what is known geologically as the Keweenawan, a series of melaphyres (old basic lava flows) called "traps," interbedded with conglomerate or sandstone and other more acid rocks of intrusive origin. The beds outcrop at points along the central portion of the peninsula, and dip at varying angles in a northwesterly direction under Lake Superior, being steeper at the southern end and gradually getting flatter at the northern, outcropping again on the north shore of the lake. The width of this series of flows varies from 5 miles, at the northern point, to four times that width at the extreme southwest end. A great fault, running generally parallel with the peninsula and along its entire length, dipping presumably slightly to the northwest, cuts off these flows along a northeast-southwest line. This fault no doubt was the means of lifting and tilting these beds, causing them to shear and fracture. It is the opinion of some geologists that this fault was the means of bringing in mineral solutions (from a deep-seated source) from which the copper and other minerals were deposited, these sheared and fractured zones giving the solutions access to the porous and shattered beds.

Topography, Climate, Etc.

The mines of the Copper Range Co. are situated from 600 to 750 ft. above Lake Superior, which is 600 ft. above sea level.

The climate is not severe, the winter temperature averaging around 15° F., with some days as low as 20° below zero. The summer temperature is around 65°. The annual precipitation is about 33 in. Snow covers the ground about 5 months.

Forest products for underground purposes are plentiful in the district, although for shaft work most of the square timber is brought from the Pacific coast. Explosives are manufactured in the district. The overburden is glacial drift, mostly clay; the topography, rolling with a general sloping moderately toward the lake, affording a free and high percentage of run off. Plenty of good water for boiler purposes is available, and the mine pumping is light.

The mines are connected by a standard-gage railroad with the stamp mills (14 miles away on the shore of Lake Superior) and the smelter at Houghton. Houghton is the shipping point for the east, by rail or water. Coal is taken here by boat and most of the copper is shipped by boat, except during the winter.

Water power has not been developed. Labor, in addition to Americans, includes Finns, Croatians, English, Italians, Poles, Lithuanians, and Russians. These as a rule are efficient workmen. The district is not unionized.

Exploration

The location and exploration of copper deposits in this district are usually done by sinking test pits or trenching when the overburden is not too heavy; otherwise inclined holes are sunk by diamond drills, usually at right angles to the plane of the dip. The cores thus obtained, however, are used merely for correlating and showing presence of copper, and not for determining values, as the non-uniformity of the copper occurrence, even in the same bed, makes the drill core unreliable for this purpose. The copper deposits of the district are most difficult to estimate or value, for it is impossible to sample the rock in the manner generally applied elsewhere. The usual method of determining size and mineral content is by sinking a shaft in, and inclined with, the lode and extending drifts in, and carried the width of, the lode at depths of 200 or 300 ft. extending the length of the deposit. By picking out the copper rock from the rock broken (keeping separate, if desirable, that of certain sections drifted) and subjecting the former to a mill test, the values per square foot of lode thus drifted can be determined; and these sectional determinations applied to their respective intervening areas give a fairly approximate method for the determination of values and quantities. A discussion of the values and quantities of ore necessary to make a mine is beyond the scope of this paper. However, I would refer those who are interested to the paper by F. W. Denton that was presented at the 1922 meeting of the Lake Superior Mining Institute.

Mining Methods Used

At the northern end of the district, where the dip of the beds is between 35° and 40°, the open-stope, overhand method of mining was introduced. Where the hanging wall is strong, as in the Kearsarge, Osceola, and Pewabic, rock pillars were left for support; in the Conglomerate lode, however, batteries of stulls are used. There has been no change, in general, from the methods formerly used, except as depth calls for provision of additional support, and the mining must be done on a more strictly retreating basis. In only one or two instances of the above was the shrinkage system tried.

Where the dip is steeper and the beds wider, the shrinkage method was formerly used; but this has been superseded by open-stope mining or, as in the case of the Copper Range mines, by the filled-stope, using the horizontal cut-and-fill (flat-back) method, with hand sorting of the broken rock in stopes. The rejected rock, supplemented by waste from development, rock broken from the sides in stopes or tailings sands, is used for fill. By the shrinkage method (also the open stope) all the vein rock broken was shipped as ore (copper rock), and not infrequently this ore, when drawn from the stopes, was subject to dilution by the waste from the hanging wall, which would fall away as the ore supporting it was removed, resulting in a yield of only 14 to 17 lb. per ton. Much difficulty was also experienced at the chutes with these large pieces, making the operation almost hopeless.

The open-stope method is most generally used. All the rock broken is hoisted as ore (copper rock), except in a very few instances where a small discard may be made underground or after it reaches the surface. To prevent the grade, or yield, getting too low, the only alternative is to exercise care not to break rock unless it shows copper.

Mining Methods of Copper Range Co.

At the southern end of the district, where the dip is from 50° to 70° and where the lodes are worked to a greater width, the filled stope is more generally used. This method, as used at the properties of the Copper Range Co., is as follows:

The three mines operate on presumably the same bed for a distance of about 3½ miles. This is one of the lower flows, known locally as the Baltic lode, and is close to the great fault referred to. This bed outcrops almost the entire length at the Baltic mine and at several points at Champion, but is covered by glacial drift at other points, which at the southern end of Champion reaches a depth of 200 ft. The strike is northeast and southwest, dipping about 70° to the northwest and flattening slightly with depth. Mining has reached a depth, at one of the properties, of 4000 ft., the average being 3000 feet.

The mineral is native copper, which occurs in irregular masses varying in size from minute flakes to pieces weighing many tons. It is found in the network of seams and along fractures (often extending far into the foot) and in isolated masses throughout the amygdaloidal capping of a melaphyre bed varying in thickness from 100 to 150 ft. The copper-bearing zone varies in width up to 90 ft., and is worked to average widths that range from 16 to 26 ft., the narrowest being at Trimountain and the widest at Champion.[1] The distribution of the copper is not uniform, but occurs in irregular patches with intervening barren areas dispersed throughout the lode; some of these are 1000 ft. or more in length or depth. Of the grades or sizes of copper produced:

6 per cent. is in masses of 20 lb. of more,
9 per cent. is in masses under 20 lb., but over ½ in. in size,
22 per cent. is from ½ to ¼ in.,
13 per cent. is from ¼ in. to 20 mesh,
25 per cent. is from 20 to 60 mesh,
25 per cent. is below 60 mesh.

The lode itself is strong, having been worked unsupported (when not crushed) to widths up to 80 ft. The vein rock is medium hard, but varies considerably. It is not friable, but breaks in large pieces. The hanging and foot rock are harder, and the former more friable. The weight of the rock varies considerably, this being, roughly, rock in place, from 11 to 12 cu. ft. per ton.

The copper content of the lode is not the same at the three mines. Records of production show that the lean mine averaged only 21 lb. per sq. ft. of lode mined, while the richest averaged 44 lb. The stamp rock (selected ore) produced per square foot of lode mined for the former was 0.98 ton, and for the latter 1.36 tons; while the lode area that produced ore was, in the former, 48 per cent. and, in the latter, 66 per cent. of the total area opened. The method of grading is by classifying (by eye) monthly into "good," "fair" and "poor" the openings made, the good and fair being considered as stoping ground. Such classification gave averages of 40 and 61 per cent., respectively, or slightly under the actual.

We have no figures from which to derive the ratio of waste mined to ore for the same period. This, however, will be discussed later under "Sorting."

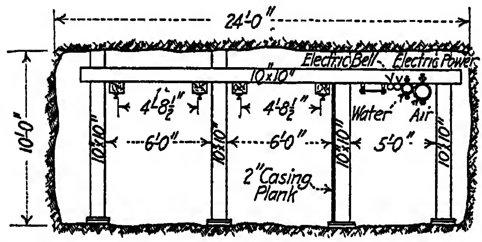
The rock temperature at 4000 ft. is about 75°, diminishing, roughly, 1° with every 100 ft. of rise. No gases are generated to our knowledge,

[1] In contrast with these widths, the mines at the northern end of the district, except only the C. & H. conglomerate, work to a maximum average width of 12 ft. The Kearsarge and Osceola lodes are worked from 9 to 12 ft., the Pewabic only 5 ft., while the Conglomerate referred to is worked to about 15 ft.

and natural ventilation has sufficed, except in long drifts, where forced draft is used.

Shafts

The workings are reached by shafts inclined at the dip of the lode and sunk either in or out of the lode, spaced from 1000 to 1500 ft. apart. The present depths of the shafts along the incline range from 2500 to 4000 ft. The shafts have three compartments: two, 6 by 7 ft. for skip roads, and a 5 by 7 ft. for ladder road (between timbers). Where shafts go through the overburden, the construction is of concrete; where in



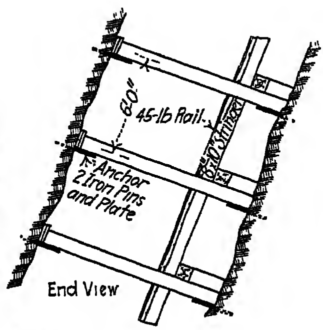
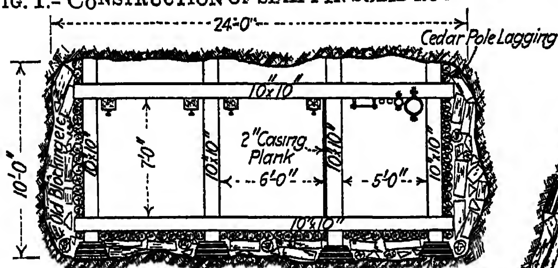


FIG. 1.—CONSTRUCTION OF SHAFT IN SOLID ROCK.



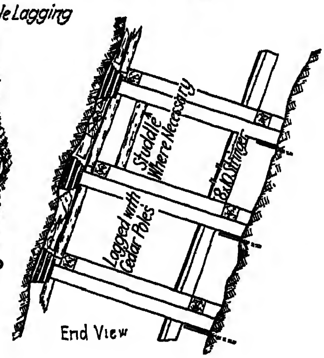


FIG. 2.—CONSTRUCTION OF SHAFT IN CRUSHED GROUND.

rock, timber is used. In solid rock, the construction is as shown in Fig. 1; in crushed ground, as shown in Fig. 2.

Hoisting is done in balance and from all levels, using 5-ton skips. The rock is dumped directly from 2½-ton, end-dump cars into the skip. Dump cradles used are so designed that the inertia of the moving car, when it strikes the cradle, tilts the car to the dumping position.

Shaft pillars were originally left in the stopes over, under, or at either side of the shaft, depending on the shaft's location, and were at first made 100 ft. wide, but later were increased to 200 ft. When the average stoping depth had reached 1500 ft., and as the area stoped at this depth exceeded 50 per cent. of the lode area, the crushing of these pillars fol-

lowed, causing much trouble and considerable expense to maintain the shaft roads. Where possible, these pillars have since been removed, giving relief to the shafts; and present practice is to stope through without leaving pillars. Sufficient ground is left between the stope and the shaft to protect it from a possible breaking through of the fill. Where the shaft is in the lode, a narrow stope is carried in the foot or the hanging

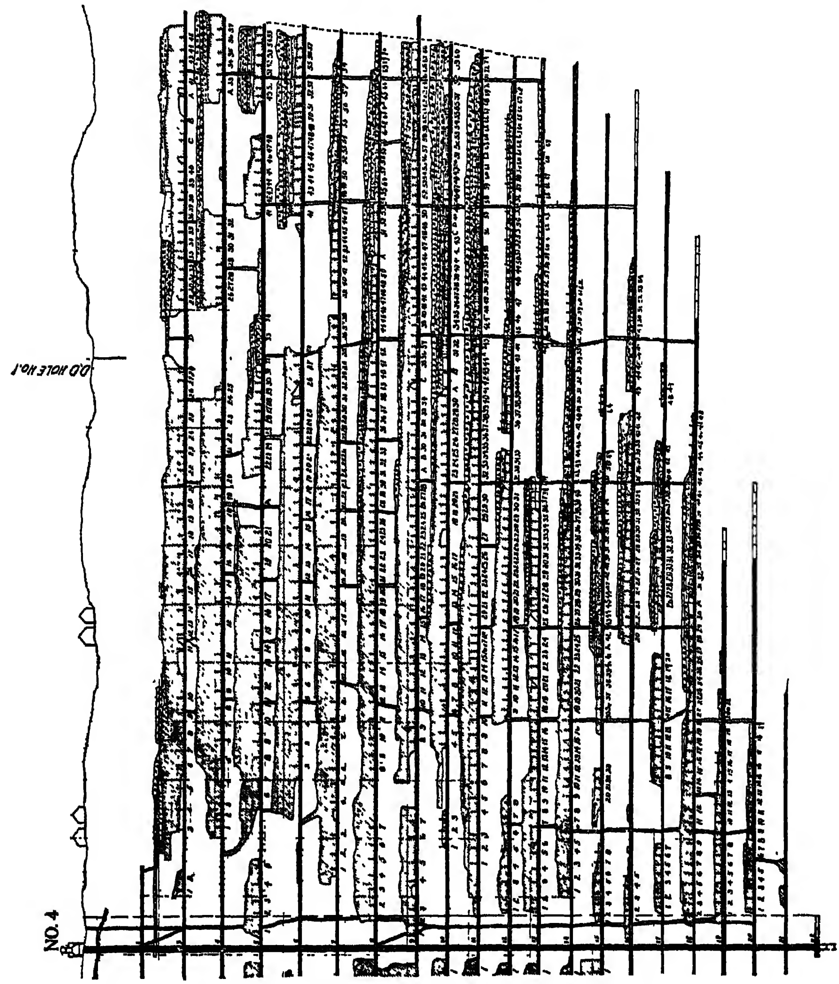


FIG. 3.—SOUTHERN END OF CHAMPION MINE, SHOWING CONTINUOUS STOPING

trap, under or over the shaft, to connect the main stopes, thus permitting the distribution of the weight of the hanging country to the fill, instead of concentrating it on the shaft pillars.

Development

Working levels are opened the full width of the lode at intervals of 100 ft. along the dip. At intervals of 400 to 500 ft. along the levels, as

the drifts advance, raises are put through (in the lode) to connect with the levels above for ventilation; and later to be used to pass fill or supplies down into the stopes. These raises are placed one under the other to permit passing fill down several levels without necessitating a transfer. Another series of raises extending from surface and located close to the shaft and at several points connected with it, is used to bring sand into the mine for fill and for its distribution to the levels; it is also used to pass waste rock hoisted in the shaft to these levels. A section taken in the plane of the lode, showing arrangement of levels, raises and stopes, is shown in Fig. 3.

When a drift has advanced 500 ft. or more, the back is taken down to a height of 16 ft. This is called the "cutting-out" stope. The broken

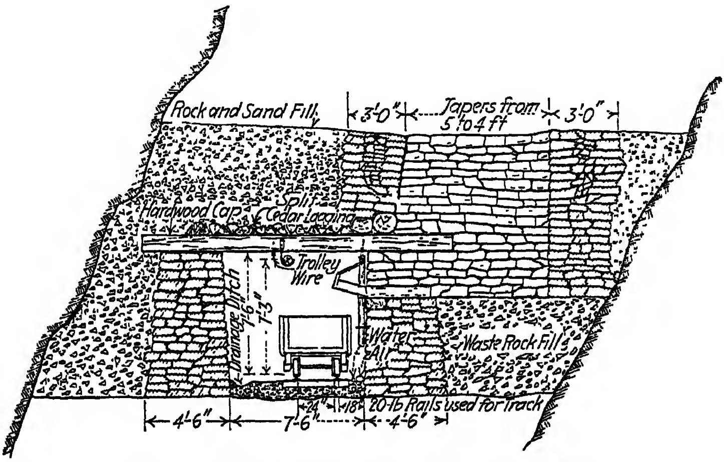


FIG. 4.—SECTION THROUGH LEVEL AT CHUTE.

rock is hand sorted, and the larger pieces containing no visible copper are used in building dry walls, which form the sides of the gangway or haulage level. The rock has a tendency to break in tabular pieces well suited to this work. At 25- to 30-ft. intervals, openings are provided for chutes, or mills (usually built on the foot side of the level) which also are built of rock, made circular and carried up through the fill to the stope, to serve as ore passages or ladder roads. A section through a level at one of these chutes is shown in Fig. 4.

The walls of the gangway are covered with hardwood caps and split cedar lagging; 30-lb. rails are used and the track is laid to a grade of 0.25 per cent., with a gage of 24 in. Most tracks are laid straight, with only slight curves. Where water is conveyed on levels, a shallow ditch is provided on the hanging side. Compressed-air lines on levels graduate so that the last 1000 ft. from the shaft is 2½ in. in diameter, the next 1000 ft. nearer to the shaft being 3 in., and the next 4 in.

Mining

Drilling is done with one-man hammer drills of both wet and dry types, equipped with tappet chucks and using either 1-in. or 7/8-in. hexagon or quarter-octagon steel. The bits used are of the cross or double Carr type with 1/16-in. gage reduction with each 22-in. change. Starters are 17/16-in ; and steels up to 12-ft. lengths are used.

Drifts are driven the full width of the lode (10 to 30 ft.) and 10 ft. high. The side cut is used and holes are drilled up to 10 ft. deep, using three to four holes for each cut, with a burden of about 18 to 24 in. From 20 to 40 holes are required to square a cut, depending on the width of drift carried, tightness of ground, and copper present. The heavier, mounted, water hammer drills, using 1-in. steel, are used in the drifts.

In the stopes for horizontal backs, the same kind of machine is used. A breast one hole deep, the entire width of the lode, with a 4- to 5-ft. burden, is carried. Holes are spaced about 5-ft. centers across the lode and run to depths of 7 to 10 ft. The charging and firing is done by the miners. In the stopes, for mining and block holing, 1 in., 30 per cent. ammonia powder is used; and in the drifts, 40 per cent.

The stoper type machine is used for raising and inclined stoping, and jackhamers for block holing and shaft sinking

Before regular stoping is started, the spaces behind the walls are filled with waste rock either from the cutting out or from some drift above brought in through the nearest raise. When the fill has been leveled to the top of the timber covering the walls, the first stope is taken, carrying a 2- to 3-ft. breast, so as not to break down the level timbers, which are temporarily shored up from the floor of the level, while this cut is blasted. The broken rock is sorted and the waste, supplemented if necessary with waste from the levels above, left to cover the timber to a depth of 3 ft. or more, which is sufficient to protect the level timber from subsequent blasting.

Horizontal Cut and Fill Stoping

In the regular stoping, work is begun from one of the raises, and when a pile 50 ft. or more in length has been broken, pickers are started at sorting, following at a convenient distance behind the miners. A stope car, operating on track laid on fill, is used by the pickers when the face of the pile is too far from the mill for direct handling of the rock. The waste rock is thrown or piled behind the pickers and to the side of the track. Rock is usually not trammed in the stope for more than 30 or 40 feet.

As the miners and pickers are working in one direction from the mill, fillers, starting from the same raise, work in the opposite direction. To

fill the stope, enough waste is first run through the raise to allow the pile to cone until it reaches the back. It is then leveled off to the height of the new fill. If waste rock is to be used as fill, a chute is installed under the raise for the purpose of drawing fill into the car. A regular stope car is used for this purpose, operating on track of 3-ft. gage, which is extended or trestled out a short distance beyond the pile; spreading of fill is done by dumping to either side of the track and by shifting the track as found necessary.

At the No. 4 end of the Champion mine, the development and stope rejection does not supply enough waste rock to care for all the filling. Mill tailings, or stamp sand, are hauled back and dumped into the mine to meet the requirements. Where sand is used for filling, instead of

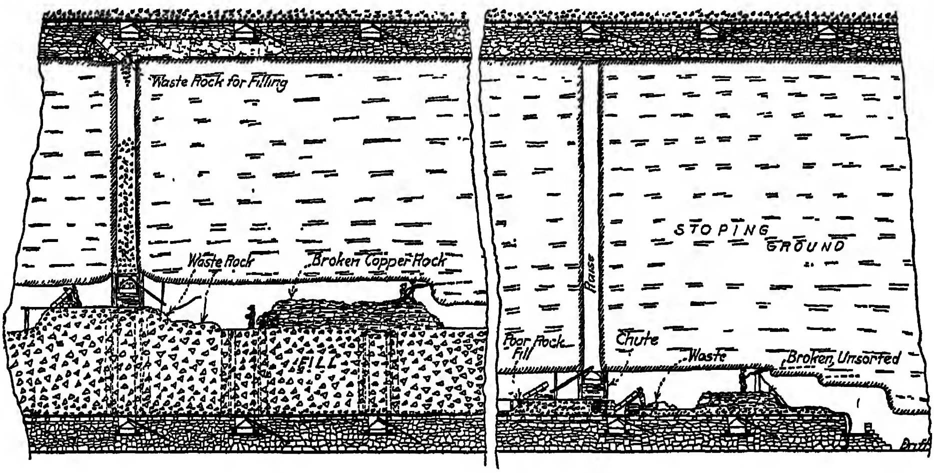


FIG. 5.—HORIZONTAL CUT AND FILL.

installing the chute, a sand tank is used and the sand spread as described later. The wallers work in advance of the fillers, building the walls to the height of the new fill.

No sand is used in a stope until two or three backs have been stoped over the level and the fill built up of waste rock to prevent possible breaking through of sand into the level or gangway. Fig. 5 illustrates the horizontal cut-and-fill method.

Sorting

In sorting, two men work together (locally called pickers). Most of the copper rock is handled directly into the chutes, but where the stope is wide or the chute spacing too great for direct handling a stope car or wheelbarrow is used.

A 1-ton car is used in a stope. This is open at one end, mounted on a low turntable truck, and hinged for either end or side dumping. It is of knockdown construction and can be taken into a stope through the chute, being easily handled by two men. The chief objects of its design were portability, low open-end construction, permitting easy shoveling, and adaptability for dumping to either side or end up to the level of the track.

The rock, as broken, lies in piles 6 to 7 ft. deep. The piles are handled as found necessary to pick out the copper-carrying rock, usually requiring the handling of the entire pile, the larger pieces by hand and the fines with shovel. The larger rocks are block holed. Any pieces of 50 lb. or less, though showing only a speck of copper, are sorted out as ore (copper rock) and sent to the mill; that containing no visible copper is thrown back into the waste pile, which follows closely behind the pickers and is carried the width of the stope, but with an opening through it for the tram road to the mill.

Sections of piles showing little or no copper are not handled unless necessary to make roadway to mill. When most of the larger pieces contain copper, the fine rock is shoveled in as copper without sorting; when the larger pieces are poor, the fine rock also is rejected.

The character of the back, or place from which rock is blasted, is often used as a guide in deciding as to whether the rock is to go as waste or ore, especially where piles were blasted for some time and are covered with dust, making sorting difficult. The rock is picked as soon as possible after it is broken, while faces are fresh and clean. By breaking one hole high, only a small amount of fines and dust is made. Careful sorting takes time, the tonnage handled being less than for straight-away shoveling.

In general, the character of the place influences the degree of picking to be done, which is decided by the shift boss. Careful supervision by him is essential, and visits are made to each working plaec as often as possible, usually two to four times each shift. There are employed under each boss from ten to sixteen pickers, besides those working on the levels.

Because of the free and continuous stoping, at times there are 50 to 100 ft. or more where piles contain little or no copper. These are not picked, but are leveled down to the height of the next fill, moving the surplus waste to adjacent stopes where fill is needed or, occasionally, to one of the lower levels.

Foot-wall side explorations in stopes also produce considerable waste, most of which is not handled, but allowed to remain for fill.

The operations of the last three years at Champion indicate that of all the rock broken in stopes, including that by regular stoping, breaking of poor backs, and from foot-wall explorations, about 46 per cent. was sorted out as copper rock (ore); 53 per cent. was used for fill in

stopes; and 2 per cent. was hoisted to surface to fill caves. Of the rock broken in stopes, 72 per cent. was handled by the pickers, 65 per cent. of which was sorted out as copper rock (ore).

Of the total fill used, 43 per cent. was rejected by the pickers; 15 per cent. was from foot-wall explorations and was not handled; 15 per cent. was from poor backs and was not handled; 12 per cent. from foot and backs, but was handled by fillers; 9 per cent. was from development, and was handled by fillers; and 6 per cent. was sand fill dumped into the mine and handled by fillers. The total tons of fill used, including sand, was 60 per cent. of the number of tons broken.

Much stress is put on proper sorting. It furnishes the cheapest fill and avoids unnecessary expense on waste rock that would otherwise be sent to the stamp mill.

Roughly, for each ton stamped one ton of waste was used for fill. Where fill is obtained outside of stopes, or where sand is used, the cost is about 50 cents per ton. Each ton of waste rejected by the pickers, so long as the stope needs fill, has, therefore, a value of 50 cents. The loss due to tramming, hoisting, transporting, and stamping a ton of waste rock is 85 cents, making a combined loss on each ton of waste stamped of $1.35. The present cost per ton of ore stamped is $2.60. Therefore, to dilute a ton of copper rock by adding a half ton of waste would in effect add 67 cents to the cost of producing the copper contained in the original ton of ore, which is an increase of 26 per cent.; and if instead, 1 ton of waste is added the cost is increased 52 per cent.

Further expansion of the continuous and wider breaking in the stopes and closer sorting have been the chief developments and improvements of the system, with a consequent finding of more copper and an increased output per underground shift as well as an increased output for the mine. This is brought out forcibly by a comparison of the results of the last seven years with those of the seven-year period just previous, which leaves no doubt in our minds as to the advantages of more complete sorting out of the waste rock and a systematic exploration for the finding of additional copper. During the second period, however, the full effect was had of the one-man water drill, sand fill, and regrinding, which were introduced during the last few years of the first period. Credit must be given for their influence, which, it is roughly estimated, accounts for about 20 per cent. of this improvement.

In addition to these advantages, there was a saving by not hoisting, transporting, and stamping the waste rock. As shown, the shaft and mill capacities for copper output were increased, also the copper produced per square foot of lode mined.

The mill operation was benefited by stamping rock of higher yield; for additional treatment was possible, which would not have been the case had the yield been low. As a result, complete regrinding and table

TABLE 1.—*Comparison of Results of Last Seven Years and Those of Seven-year Period Just Previous*

(Figures are Average for Each Seven-year Period)

	1908–1914	1915–1921	Increase, per cent.
Yield of copper per ton	24.8	37.2	50.0
Copper produced per underground shift	63.5	99.5	56.7
Powder used per ton ore stamped, pounds	1.13	0.94	44.0[a]
Compressed air per ton of ore, cubic feet	3,270	3,290	36.0[b]
Average yearly production of copper, pounds	16,500,000	24,300,000	46.0

[a] Decreased consumption per pound copper produced.
[b] Decreased consumption per pound copper produced.

treatment were added, giving an additional recovery from the tailings equal to 4¼ lb. of copper per ton stamped. Or stating it in other words, the copper lost per ton of mill tailings from the higher-yield rock is now no more than that from the lower; and as 50 per cent. more of the lower-yield rock had to be shipped to produce the same amount of copper, the mill tailings loss from this additional tonnage is eliminated. Against these advantages must be charged part of the copper lost in the fill. What this is, we do not know, but it is not enough to pay for its treatment.

Exploration in Stopes

Before starting regular stoping, and subsequently at intervals as the stope is carried up, a thorough inspection of the foot wall is made and the stope widened to the probable limits of copper. The rock broken this way is left for fill; where copper is uncovered, it is followed by subsequent cuts. Many isolated patches of copper are found in this manner, the work having proved very profitable. Large associate bodies have also been discovered; one notable for its size and richness, extending 1000 ft., in length and depth, lies parallel to the lode and about 150 ft. to the foot of it, but communicates with it at its upper limit, being mineralized from 10 to 40 ft. wide.

To aid in the finding of these foot deposits, horizontal sections of the stopes are made at 20-ft. intervals. These are reproduced in plan to convenient scale on glass plates, which are mounted in a rack superimposed and aligned to correspond with the relative positions of these stopes, thus making a transparent model showing the relation of the workings in levels and stopes at a glance. If a stope directly under a level shows wide, it indicates that a search for its extension should be made on the level, or vice versa. A careful mapping of the more pronounced slips is also made.

Inclined Cut-and-fill Stoping, or Caving of Level Pillars

When a stope has been worked by the horizontal cut-and-fill method to 80 ft. above the level, or to within 20 ft., of the level above, the floor pillars are extracted by starting at a raise midway between shafts, or at the outside limit of the stope, as the case may be, and an inclined stope formed from the bottom of the raise, running first sufficient fill through the raise to permit miners to reach the back, and cuts are then inclined to the slope of the fill.

The pickers sort the rock as it is broken from day to day, working on the slope of the fill. The copper rock is thrown or rolled to the foot of the

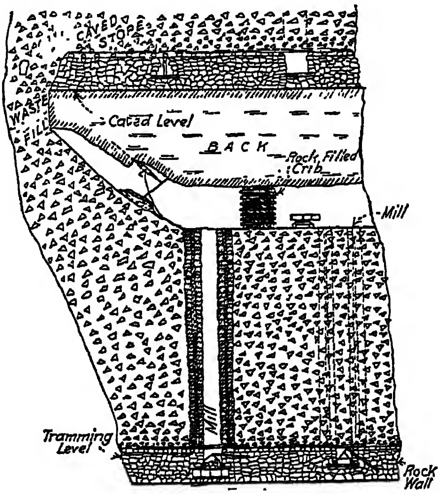


FIG. 6.—SECTION THROUGH CAVED STOPE.

pile and then into the car or chute. The poor rock is left on the slope for fill. A deep breast is avoided so as not to break the rock into large blocks, as these do not roll or slide readily, but tend to stand at a steep angle, making conditions dangerous for pickers. A one-hole breast stope inclined to slope with a 2½- to 3½-ft. burden is carried. Hand rotated stopers with ⅞-in., quarter-octagon, solid steel are used for this work, one miner breaking sufficient rock to keep two pickers busy. When the cut has been completed and the broken rock picked over, fill is again run through the raise until the pile is brought up so miners can take the next cut, then repeating the operations until the slope of the fill reaches the collar of the raise. This opening is then widened to the width of the stope, causing walls and level timber to cave.

The next cut on the face of the pillar is carried up within 6 or 7 ft. of the floor, leaving a horizontal stub at the upper end of the pillar, which is drilled over but not blasted until the broken rock of the last cut has been

removed or picked over, the sloping face of the stope being kept open by this stub. When the last cut has been picked over the stub of the pillar is blasted, allowing the level timber and fill from the level above to cave, and the fill to advance, repeating alternately this operation with the breaking and sorting until the end of the stope is reached. Some of the rock from the stub end of the pillar is lost in the fill, but if blasting is done in stages, this loss can be greatly reduced. When the cave has been worked back a sufficient distance, caving on the level below is started, and so on down, the series making an inclined retreating line as it approaches the shaft.

As the upper levels are being caved and the fill drawn down, waste from the developments or sand is dumped into the mine at surface to replace the fill as it recedes. A section through cave is shown in Fig. 6.

Working and Recovery of Crushed Stopes

The crushing of shaft pillars is usually accompanied by crushing of the floor pillars. When 50 per cent. of the lode has been extracted from an area 1200 ft. or more in length, and the mean hoisting depth of that area has reached about 1500 ft., crushing takes place. This crushing is gradual, affecting first the thinnest pillars, and warning is given in sufficient time to permit the placing of supports to prevent falling in of these backs. Crushing usually starts with the removal of the floor pillar. Little or no timber is required until the stope has been carried up 70 or 80 ft. As the stoping along the levels is progressive, the backs farthest from the shaft are the thinnest; and as the work on the upper levels is progressively in advance of that on the lower, the crushing is localized to the area near the caves.

When the stopes have been carried to a height ready for caving, timber cribs or bulkheads are built of hardwood pieces 8 ft. long and 8 to 10 in. in diameter and filled with waste rock. They are spaced about 25-ft. centers and are wedged tightly against the back. These cribs, or rock packs, are supplemented with occasional props placed between them where necessary. Large footings 3 or 4 ft. square are provided for the larger props. The smaller ones are supported on large, flat rocks or short pieces of ties. In crushing, the rock breaks into large pieces which, when propped, help to support the overlying pieces. The stope in this way can be kept open until the pillars can be caved.

In very wide stopes, the lode is not removed the full width on the last two cuts. The foot side is left standing to reduce the width of the back and to help support the pillar. This ground is later removed to the width of the stope by vertical cuts, taken one at a time, in advance of the caving cut.

The crib at the foot of the cave is held as long as possible. If there is any danger of the brow not standing with its removal, a crib built farther back, supplemented with props, is used.

Timber in the cribs, as a rule, can be used several times. The building of these cribs, which are about 7 ft. high, cost last year for labor and supplies about $25 each, or approximately 10 cents per ton of stamp rock supported.

When crushing of the floor pillars first started, thousands of feet of backs crushed and caved before supports could be introduced. These backs would work up in the form of an arch; the loose material falling away would almost entirely fill the stope. The backs, though crushed, would stand fairly well when thus arched. All the mills in these stopes would be covered with 10 to 25 ft. of loose rock.

These crushed backs, as well as the broken rock lying in the stopes, were later recovered. Access to a stope was made through a raise from the level above, the back carefully barred around the raise, and the rock over the mill directly under the raise blasted from underneath and run through the mill. When the pile of broken rock was drawn down to the floor, or fill, of the stope a working level was established. Pickers were put at handling the rock on this floor into the mill, working in the direction of the next mill. As soon as space was made, a dry-wall gangway, similar to that on the level but smaller, was built, making thus a sublevel connecting with the mills of the old stopes as they were recovered.

Chute or mill connections were also provided in the wall to the stope about to be formed above the sublevel. After 15 or 20 ft. of the walls were covered with timber, fill was introduced through the raise to cover this timber to a height sufficient to furnish head room under the back, forming thus a new stope floor, on which cribs or bulkheads were built to support the back.

The fallen rock was then handled from the floor of the sublevel, the men working under cover of the timber and fill, which were kept close to the working face of the pile. The back over the working place could be reached and barred from the fill as it advanced, making the place reasonably safe. The timber cribs and props also were kept close to the edge of the fill.

A temporary cover of 6- or 8-in. cedar poles was placed over men at the working place as an additional protection from small pieces, these poles reaching from the end timber of the sublevel to the face of the copper rock pile. Blasting of the larger rocks was necessary, the temporary cover being removed while blasting. The building of walls could also be carried on under this cover. Usually a miner, a waller and a picker were employed in each place, doing their own timbering, walling, filling and cribbing.

The mills through the new fill were raised as the slope of the fill advanced. These mills served as connections to the sublevels through

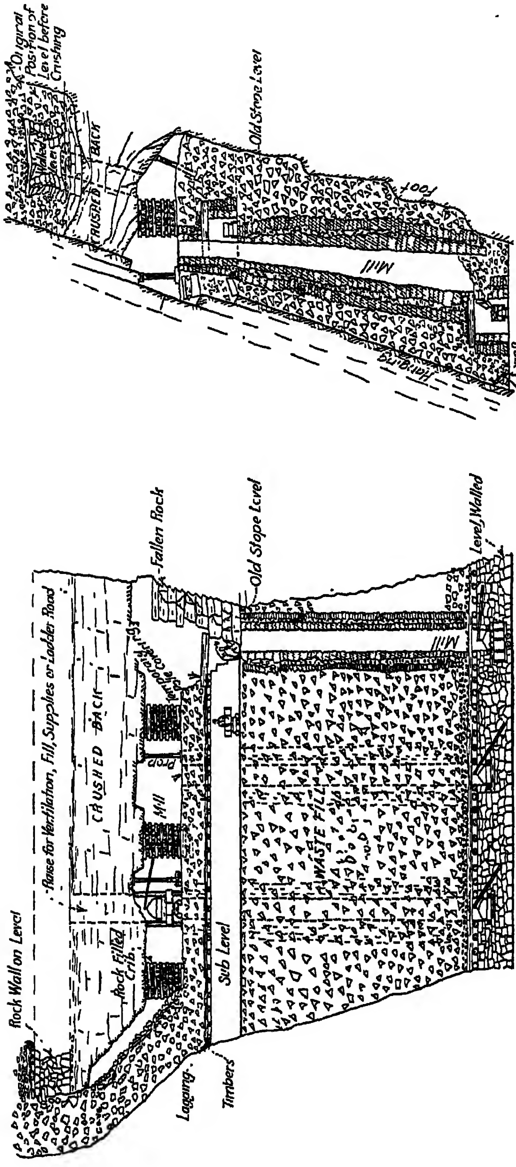


FIG. 7.—LONGITUDINAL SECTION THROUGH CRUSHED STOPE.
FIG. 8.—CROSS-SECTION THROUGH CRUSHED STOPE.

which copper rock from the back of the floor pillar could later be handled. Longitudinal and cross-sections of one of these stopes are shown in Figs. 7 and 8.

Sand Filling

There are stopes at the Champion mine where the waste from exploration, development, and sorting is not sufficient to supply the fill requirements. Mill tailings, or stamp sand, are hauled back during the summer months in 40-ton, bottom-dump cars and dumped into a raise paralleling No. 4 shaft that connects with all working levels. The sand is drawn from chutes in 1½-ton, bottom-dump cars, usually six at a time, and hauled by motor to raises communicating with the stopes below, using raises as storage to feed sand blowing tanks operated by compressed air and located directly underneath the raise. The bottom of the raise is stopped off by a sollar, to which is attached a 10-in. pipe and gate for drawing sand from the raise.

The tank is cylindrical and has a capacity of about 1½ tons. It is coned at both ends. The top cone contains a counter-balanced gate that swings down away from the flow of the sand when the tank is being charged. This gate, or valve, seats against the under side of the head by means of a rubber gasket, thus making an air-tight compartment when closed. The bottom cone of the tank connects to a 6-in. tee placed horizontally and connected at one end through a reducer to a 4-in. distributing pipe which rests on the fill. The other end of the tee has a reducing flange, to the inside of which is secured a short 1-in. pipe nipple, which is reduced at the end to form a nozzle. A 1¼-in. pipe connects the outside of the reducing flange to the top of the tank, and in turn, is connected to the main air line, valves being suitably placed to permit regulation of air to either the top or the bottom of the tank as required.

Extra-heavy black pipe, cut to 6-ft. lengths and joined by Dresser couplings, is used for the distributing line. These couplings allow deflecting the pipe around obstructions. If the pipes were threaded only about one-half of the wear would be obtained, for when the pipe would be half worn the root of the threads would be exposed and the thread cut off, making it necessary to discard the pipe. But with the couplings, the pipe can be used until worn through; frequent turning of the pipe distributes the wear. The pipe lasts from 3 to 6 months.

The sand is drawn from the raise into the tank; when the tank is almost full the upper gate is closed, and then the counter-balanced valve. Compressed air is applied, which seats and holds the balanced valve, and the pressure in the tank feeds the sand through the lower cone into the tee. The air jet from the nozzle cuts away the sand as it falls and carries it along through the distributing pipe. When the tank is nearly empty the air is shut off, allowing that which is confined in the tank to expand and expel the remainder of the charge. The upper valves are again opened, the tank filled, and the operation repeated. By shifting the line, the sand filling is carried the width of the stope. As the pile of

fill advances, the line is extended. The sand leaves the pipe with sufficient force to be carried about 20 ft. beyond the pipe.

Sand is transmitted up to 250 ft. without difficulty in this manner. The air pressure used is from 70 to 80 lb. To load and discharge a tank of sand (1½ tons) requires from 2 to 3 min., depending on the length of

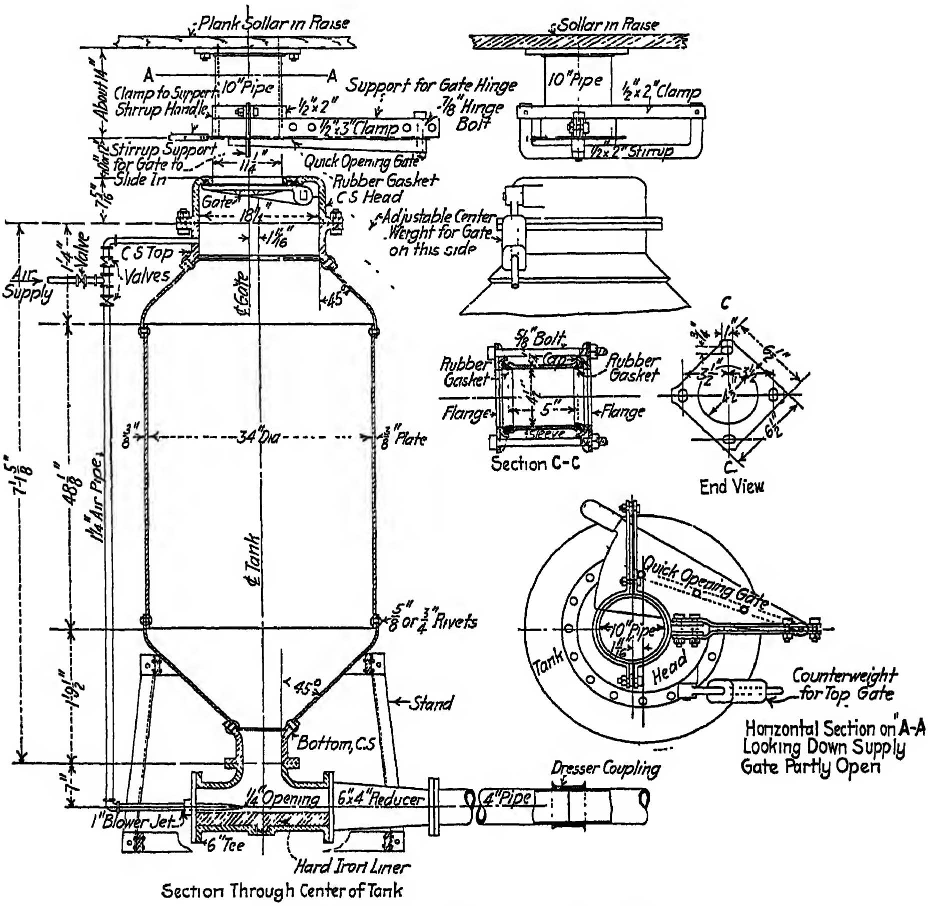


FIG. 9.—SAND TANK AND EQUIPMENT.

the line, fineness of sand, and moisture contained. The shorter the line, or the coarser the sand, up to quarter-mesh, or the drier it is, the greater the speed of operation.

One man is required to operate the tank and extend the line. From 500 to 1500 cu. ft. of free air is used per ton of sand moved. With coal at $6, compressed air costs 0.2 cents per 100 cu. ft., or 1 to 3 cents, with a probable average of 2 cents, per ton of sand handled. Fig. 9 illustrates sand tank and equipment used.

Timbering

The only square timber (which is fir) is used in shafts. In drifts, 12- to 16-in. round hardwood pieces 14 ft. long are used for caps, covered by split cedar lagging in 16-ft. lengths. As a rule, only after a stope has been carried up to caving height is it necessary to use timber to support the back. For this purpose cribs are built of round hardwood pieces from 8 to 10 in. diameter and 8 ft. long. The little framing necessary is done underground. Much of this timber is recovered and reused. The timber used in stopes is delivered by motor to the nearest raise and lowered or dropped into the stope. No sampling is done either underground or of the product on surface.

Loading Machines and Scrapers

A No. 0 Thew electric shovel, in conjunction with an apron conveyor, also a Myers-Whaley and several types of scrapers have been tried. The use of the latter two has been discontinued, as sorting cannot be properly done in either case and the large rocks caused much trouble in their operation.

Underground Haulage and Tramming

At the Champion mine, nearly all the tramming is done by motor. Trolley-type electric locomotives are used and operated on 250-volt, direct current, and on 24-in. gage tracks of 30-lb. rail (laid on 0.25 per cent. grade) bonded on one side. No. 00 trolley wire is used, placed 7¼ ft. above the rail and 2 ft. to the side of the center line of track on the opposite side from the chutes. Wire is guarded for about 10 ft. opposite chutes to prevent accidents. Four-ton locomotives are used, of both General Electric and Goodman makes.

On the levels, 2½-ton end-dump cars are used. They have no doors and are open at one end, with bottoms about 15 in. above rail for easy shoveling from cutting outs and drifts. The rock from stopes is drawn through the chutes of the mills, and the larger pieces are used to stop off the open ends of the cars.

The cost per ton for electric tramming is about 10 cents, including all labor (power at 1½ cents per kw.), maintenance, supplies, and depreciation of equipment. Tons per man by motor haulage is more than double that by hand tramming.

Hoisting

All hoisting is done in balance. Nordberg duplex steam hoists, with composite or cone-end drums, of two sizes are used; viz., 24 by 60 in.,

with 3000 ft. of 1¼-in. rope capacity, and 30 by 72 in. with 5000 ft. of 1¼-in. rope capacity. Standard 6-strand, 19-wire, hemp-center, 1¼-in., plow-steel ropes are used.

All hoists are equipped with Lilly hoist controllers. Hoisting under our conditions, capacities of 120 tons per hour from 2000-ft. depths have been obtained. The hoisting speed for rock is 1500 ft., for men 600 ft. per minute.

Where steam is required for heating, the exhaust from hoists is used for this purpose[2] by interposing a steam regenerator between the hoist and the heating system. A total of 52,000 sq. ft. of radiation is connected in this way, obtaining about 80 per cent. of its steam from the hoists.

Pumping and Drainage

Sumps and pumps are located in one or two of the shafts of each mine at intervals of 700 to 800 ft., although one of 1600-ft. head has been lately installed. The sumps are sunk to the foot of the lode with capacities for the storage of 2 to 3 days' pumping. The average flow for the year 1921 at the Champion mine was 88½ gal. per min., with an average flow during the spring months of the year of 100 gal. Of the total, 42.5 per cent. is pumped from the 21st level, 78 per cent. from the 13th level, and the whole volume total is pumped from the 6th level. During last year 110 gal. were pumped to surface per kilowatt-hour.

On levels, water is coursed around raise openings by concrete ditches; in shafts, the water is intercepted and caught by concrete launders at points above pump levels. Except for small amounts from the bottom levels, all pumping is done by electricity, using 2300-volt, alternating current.

Mostly, vertical triplex or horizontal duplex pumps of 200- to 400-gal. capacities and for heads of 800 to 1600 ft. are used though a few centrifugal pumps of 200-gal. capacity and 800-ft. head are used. In some cases, centrifugal pumps are used as spares to the plunger pumps. Where this is not so, No. 11 Cameron pumps with special water ends are used. These, however, are used only in an emergency.

Air Compressors

A Nordberg two-stage compressor of 9000-cu. ft. capacity at 80-lb. pressure, driven by a quadruple expansion steam engine, running condensing, furnishes the air for the Champion mine. A three-stage triple expansion machine of 4700-cu. ft. capacity is used at Trimoun-

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[2] Steam Regenerators Reduce Coal Consumption. *Trans.* (1921) **66**, 636.

tain. These are each supplemented by a steam-driven machine of smaller capacity.

Ventilation

Natural ventilation is employed in stopes and drifts; and with the aid of the many raises used the stopes, as a rule, clear of smoke in 10 to 15 min. after blasting. Forced draft is used in some of the deep levels, using 2000-ft. centrifugal blowers.

Lighting

Electric illumination is used at all shaft stations on all levels using electric haulage; carbide lamps are used by individuals.

Telephone and Signal Systems

Stromberg-Carlson telephones are used on all busy levels. An electric mine signal system developed by the Copper Range Co.'s electrical department is used. The sender of a signal hears the signal repeated back by buzzers at each station. A description of this system has been published.[3]

Record of Unit Production

The unit production figures and costs here given are for the Champion mine for the year 1921. The tonnage stamped was at the rate of about 60 per cent. of what is now considered normal. Coal and transportation costs were double and other supplies about 50 per cent. above prewar averages. The ton of 2000 lb. is understood. All labor is employed by the day, except in some of the drifts where miners and trammers are on contract. The miners' contract is figured by the foot advance, with allowance for width, copper present, and hardness of the ground; the trammers', by the car.

NOTE.—Where tons per man or units per ton are given it refers to copper rock (ore); this, because of the sorting, is only 50 per cent. of the rock broken.

	Tons per Man per Hour	Man-hours per Ton
All miners in stopes and drifts (machine men only)	2.44	0.41
All men underground	0.32	3.13
Underground, surface and stamp mill, not including office	0.236	4.24
All men, including office force	0.231	4.33
All men at mine, including office	0.26	3.85

[3] *Electrical World* (Aug. 21, 1920).

Classification of Labor for Total Organization, Expressed in Percentage of Total

Underground at mine	72.3		
Surface at mine	16.8	Surface and shops to underground	4.4
		Rock-house	2.0
		Hoisting	3.8
		Compressors and drills	3.1
		Car filling	0.3
		Surface	0.8
		Office and general	1 9
		Water supply	0.5
			16.8
Stamp mill	10.9		
	100.0		

Classification of Underground Labor in Percentage of Total Underground Force

Pickers	30.0
Miners, company account	11.7
Trammers, including shovelers in drifts	11.6
Stope fillers	11.1
Timbermen and helpers	9.6
Wallers	7.3
Electric tramming	5.4
Bosses	3.2
Dumpers	2.6
Track layers	1.7
Contract miners	1.5
Block holing	1.1
Miscellaneous	3.2
	100.0

The tonnage stamped for the year was	531,780
Pounds of copper produced	20,748,934
Yield, or pounds copper produced per ton stamped	39
Pounds of copper produced per man-shift, entire payroll	72.5
Pounds of copper produced per underground man-shift	100.0

	Total	Labor	Supplies	Other
Cost per pound of copper[a]	$0.0757	$0.0413	$0.0238	$0.0105
Percentage of cost		54.6	31.4	14.0[b]

[a] Does not include smelting, eastern office, taxes or depletion, but includes transportation and stamping.

[b] Of this 14 per cent. 10½ per cent. is for transportation.

The labor turnover for the year was 160 per cent., which includes men that were rehired after drawing their time, even though their leave was for only a few days.

UNIT SUPPLIES USED PER TON OF COPPER ROCK (ORE)

Explosives (30 and 40 per cent. ammonia powder), in pounds	0.68
Fir timber (square) for shafts, in feet (b.m.)	0.215
Boards (hemlock and hard wood) in feet (b.m.)	0.32
Track ties, in feet (b.m.)	0.22
Cedar poles, in feet (b.m.)	0.04
Hard wood (round) for caps, in feet (b.m.)	0.69
Hard wood (round) cribbing, in feet (b.m.)	0.38
Cedar lagging, in cords of 512 cu. ft.	0.002
Total horsepower-hour at mine	14.19

Hoisting and heating $= \dfrac{\text{Coal used for hoisting and heating}}{\text{5 lb. per hp.-hr. (assumed)}}$ 5.40

Compressing air $= \dfrac{\text{Coal used for compressors}}{\text{2.1 per hp.-hr.}}$ 3.44

All other purposes $= \dfrac{\text{Total kw.-hr. used}}{\text{0.746 kw.-hr. per hp.-hr.}}$ 5.35

	Kilowatt-Hour
Electric haulage	0.64
Pumping	0.79
Rockhouses (crushers, etc.)	0.58
Water supply	0.89
Surface lighting	0.27
Shops and all other	0.83
	4.00 = 5.35 hp.-hr.

Cubic feet of compressed air at 80 lb. (delivered) 2400

COST OF SUPPLIES USED AT MINE, IN PERCENTAGE OF TOTAL COST OF PRODUCTION TO MINE ROCK BINS[a]

Powder, fuse and caps	5.75	
Lumber and forest products	2.47	
Coal at mine	5.73	
Electric current, at 2 cents	3.67	17.62
Sand for mine filling	1.14	
Drill machine parts	1.07	
Drill steel	1.05	
Plate and steel	0.63	
Machinery used underground	0.61	
Pipe and fittings	0.50	
Carbide	0.40	
Feed for horses	0.35	
Rail	0.26	
Lubricants and waste	0.25	
Air hose	0.22	
Tram cars and supplies	0.17	
All other supplies	2.88	9.53
Total supplies		27.18

[a] Cost of production not including stamping and transportation.

As we make no estimates of ore we cannot say what percentage is actually mined.

After ore has passed through the crushers in the rock-house, it is drawn from the bins into 60-ton railroad cars and hauled to the stamp mill, where it is stamped and put through Hardinge ball mills, thus being reduced so all will pass through 20 mesh and 30 per cent. through 200. The sand receives jig and table treatments, giving an extraction of 85 per cent. or over and a concentration ratio of 30 to 1, with mineral running about 66 per cent. copper. This is shipped to the smelter, where it is melted and refined by fire process, a small amount of only one grade receiving subsequent electrolytic refining for silver recovery.

The safety program of each mine is in charge of its superintendent, with the mining captains assisting, these men making trips underground daily.

Welfare work is conducted by a community committee. A library with reading and recreation rooms is maintained by the company; bowling alleys and other recreational provisions have also been made. Medical and surgical attention is given to the men and their families by the company's hospital medical staff. A charge of $1.50 per month is made for this service. The employees contribute 50 cents per month to an Aid Fund which pays $1 per day during the time an employee is absent, because of any injury received while at work, and from $600 to $1000 to beneficiaries upon the death of an employee, by accident or otherwise.

Mining Methods and Costs at the Iron Cap Copper Co., Copper Hill, Ariz.

By Charles E. Lees, Copper Hill, Ariz.

The Globe Mining District is in the southeast central part of Arizona, in Gila County. Globe, with a population of about 7000, is the terminus of the Arizona Eastern R.R., a branch line 130 miles long that connects with the Southern Pacific R.R. at Bowie.

In 1874, prospectors crossing the Pinal Mountains from the west located what is generally known as the Old Dominion mine. For several years, it attracted little attention, because of the greater interest aroused by the discovery of high-grade silver ores in some of the foothills northeast of Globe. About six years later, the prospector turned his attention to the abundant copper ore revealed by surface workings along the Old Dominion vein, and, in 1884, the Old Dominion company erected two 30-ton furnaces. From 1888 to 1893, the Old Dominion company is said to have maintained an average annual production of about 8,000,000 lb. of copper. Until Dec. 1, 1898, all supplies had to be freighted into Globe by wagons and the mines of the district operated intermittently because of high expenses, but with the advent of the railroad the Old Dominion company continued to be a large and steady producer.

In the Globe district, the production of copper far exceeds in importance that of any other metal. There are four operating companies along the Old Dominion vein and the total annual product of these properties for 1923 was about 45,000,000 lb. of copper.

The unit size of the mineral tracts in the district is the regulation mining claim, 600 ft. wide by 1500 ft. long, and all ownerships are held in fee.

Globe is 3600 ft. above sea level, and lies between the Apache Mountains to the east and the Pinal Mountains to the west. The principal drainage of the district is northward through Pinal creek into the Salt River. The general slope from the high point along the vein, where the Superior & Boston mine is, to Pinal creek, where the Old Dominion mines are, is about 250 ft. to the mile.

Geology

The oldest rocks in the district are pre-Cambrian crystalline schists known as the Pinal schist, which are the basement upon which all the later rocks were deposited. These latter rocks comprise shale, con-

glomerates and quartzites with a total thickness varying from 500 to 800 ft. and are thought to be Cambrian in age. Overlying these rocks is a series of limestones, known as Globe limestones, that vary in thickness from 300 to 500 ft. and range in age from Devonian to Pennsylvanian.

These rocks have been cut by numerous faults, and following or accompanying the faulting large sills and masses of diabase were intruded between the sedimentary beds. A long period of erosion followed, during which the region was deformed by further faulting to its present topography and during which the original ores were deposited.

The main fault in this district and the one along which most of the mining is carried on is known as the Old Dominion fault; it varies from 3 to 50 ft. in width and is developed for a length of approximately 3 miles. The fissure has a variety of strike and dip but is roughly northeast and southwest, with a dip of about 80° to the south.

The fault is fairly conspicuous and is easily followed, except where it is wholly in diabase, when its course is marked by a zone of brecciation stained with hematite and salts of copper.

The vein is commonly made up of brecciated shale or quartzite and mineralized with oxide of iron and the ores of copper, the overburden varying from 200 to 600 ft. The mineralogical character of the ores along the vein is simple. The oxidation of the sulfides has resulted in simple products. The pyrite and chalcopyrite have their sulfur replaced by oxygen, carbon dioxide, or silica and become hematite, limonite, cuprite, malachite, or chrysocolla. The secondary sulfides recognized in the district are chalcocite and bornite. Native gold, silver, and copper have been observed in small amounts within the zone of oxidation.

Exploration

Most of the exploration along the Old Dominion vein has been done by test pitting, tunneling, trenching, shaft-sinking, drifting, and crosscutting. All development is carefully sampled, the outline and extent of orebody carefully determined, and ore estimated on a basis of 11 cu. ft. per ton. The production, as indicated by exploitation, has proved the method sufficiently accurate.

Change in Mining Method

The principal mining method formerly in use was the square-set method but the decreasing copper content and the increasing cost of timber, together with the increasing cost of labor and supplies, made it imperative that a cheaper method be substituted. In some places along the vein where the ground is heavy and where it is imperative to keep timber close to the working face, square-setting is used, but in general that method has been superseded by newer and more economical methods;

the selection of method depends on the size and shape of the orebody and the character of vein filling and walls.

Sampling and Estimating Reserves

The method of sampling is far from elaborate; grab samples only are taken from each round blasted daily. A record is kept of all samples taken in the block of ore lying between any two raises and the numerical average for the month is assumed to represent the value of the ore mined that month.

When estimating the reserve, which is done the first of every year, the area of each block of ground remaining between any two raises is measured on the profile tracings with a planimeter. This area multiplied by its average width gives the contents in cubic feet. This figure divided by 11 (11 cu. ft. ore in place is equivalent to 1 ton) is reported as the tonnage for that block. The numerical average of the year's samples of ore broken, together with the assay values of the drift over this section, is reported as the assay value of the block remaining.

The total ore reserve is computed by multiplying the number of tons by the per cent. for each block. This product divided by the total reserve tonnage gives the average per cent., which practically checks with the heads reported by the mill. Blocks of ground that have been worked out have been found to check within 0.5 per cent. on both tonnage and value reported.

Development

The section of the Old Dominion vein along which the Iron Cap Copper Co. is mining is about 3500 ft. long, and the width of the vein varies from 3 to 40 ft. with an average dip of about 80°. The vein material is a hard brecciated quartzite or shale between fairly good walls. The distribution of values through the deposit is irregular and some sorting is resorted to. Very few waste bodies occur in the ore zone, however, and when found are usually left in place until stopes are ready for waste filling; they are then blasted down and become part of the gob.

The inclined cut-and-fill system is used throughout the mine. This method requires very little timber; stope floors are carried on an incline of about 34°, which eliminates most of the labor of shoveling but which is not so steep as to constitute a hazard from rolling boulders.

The average stope temperature is about 78°, average relative humidity 88 per cent. Production is approximately 150 tons per 8-hr. shift of 50 men. This includes all underground labor, but only about 50 per cent. of the shift are on actual stoping operations.

Fig. 1 shows the successive steps from the starting of a stope to its finish. This plan calls for a main shaft for the handling of all men and

materials and the opening up of the mine by a series of levels placed approximately 100 ft. apart.

The Iron Cap Copper Co. has two three-compartment shafts, each compartment 4½ by 5 ft. in the clear, timbered with 10 × 10-in. timber sets on 5-ft. centers and lagged with 2 by 12-in. lagging. The Iron Cap shaft, the only one at present operating, is 1540 ft. deep, and is in the hanging wall. Commencing at the 800-ft. level, stations 15 ft. high by 40 to 60 ft. long are cut every 100 ft. Station sets are 10 by 10-in.

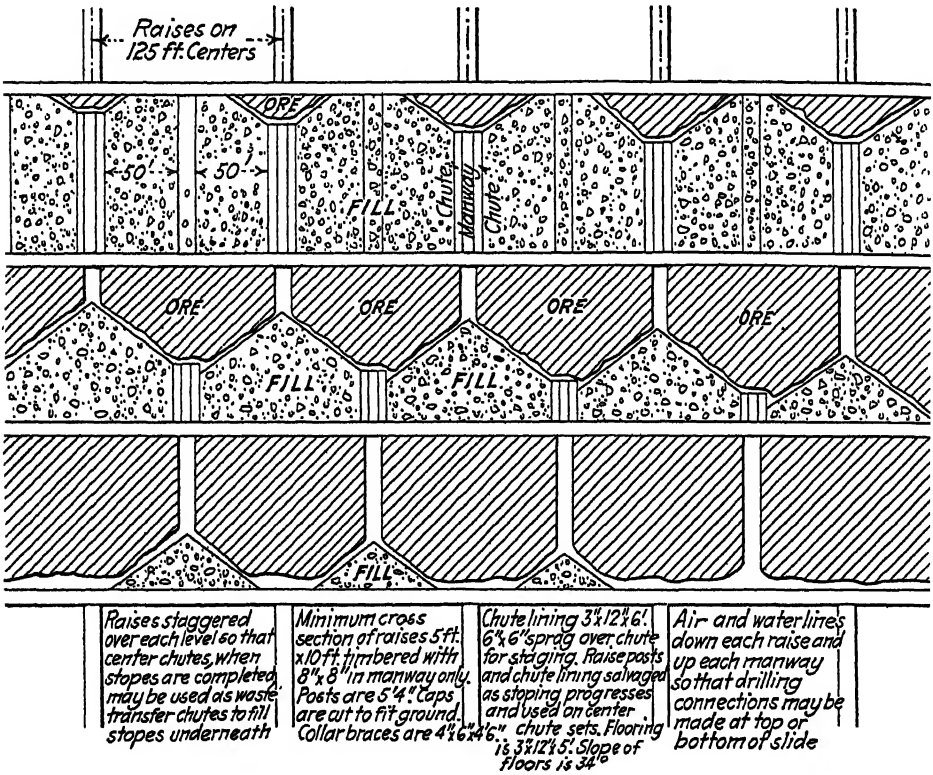


Fig. 1.

timber on 5-ft. centers with a drop of 6-in. on each set, leaving the back end of the station 11 ft. high.

Loading pockets were cut above the 1100-ft. level and under the 1300-ft. level; raises driven from each pocket accommodate the ore mined on three levels.

The shaft is so conveniently located with respect to the vein that all ground between shaft and vein constitutes the length of the station. Drifts 5 by 7 ft. are then driven along the footwall, no timber being used until stopes are started. Fig. 2 shows the arrangement and dimensions of shaft, station crosscut, and drift.

Mining Methods

As soon as the drifts have advanced far enough, raises with a minimum cross-section of 5 by 10 ft. are driven on 125-ft. centers. The sill floor sets of these raises are timbered with 10 by 10-in. timber. Posts are 8½ ft. high, sets placed on 5-ft. centers. Above the sill floor set, however, the timber is 8 by 8 in. Posts are 5 ft. 4 in. long.

The manway only is timbered with what is termed a "clap-me-down" set. The posts are set as nearly over each other as possible,

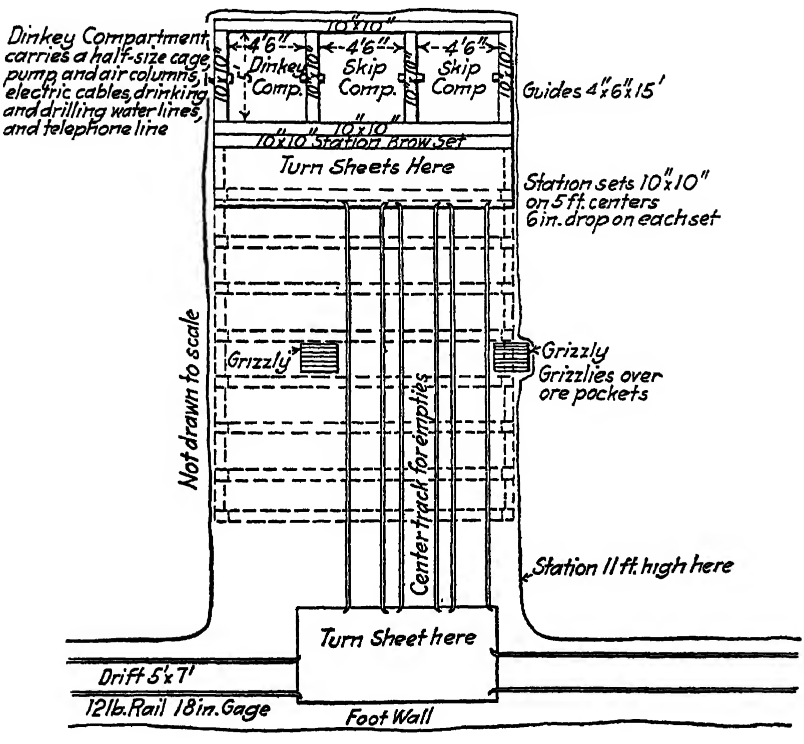


Fig. 2.

the cap is cut to fit the ground and is well blocked; 3-in. by 12-in. by 6-ft. lining boards keep the manway clean. A 6 by 6-in. sprag in the chute end flush with the last cap serves as staging for the machine men.

All headings are given definite numbers, which indicate to a certain extent their location with respect to the shaft. Headings to the east of the shaft have an even number and headings to the west, an odd number. Raises are numbered consecutively from the shaft in each direction. For instance, 902 raise No. 4 would be the fourth raise east of the shaft on the 900-ft. level. Stopes are designated as 902 stope east or west of raise No. 4.

Stopes may be started as soon as the raise has been holed through to the level above. The back of the original drift is first broken down to a height of 15 ft. and the ore mined from footwall to hanging wall. This sill-floor stope is then timbered with 10 by 10-in. sets with square framing. Sets are placed on 5-ft. centers; posts are 8½ ft. long. If the vein is not over 8 ft. wide, the cap is cut to fit the ground. In some cases where the vein is 6 ft. wide or less, and walls are exceptionally hard, hitches are cut to receive the cap and no posts are used.

A temporary chute is placed 25 ft. on each side of the original raise; permanent chutes are placed 50 ft. on each side of the original raise with a manway between.

Filling Stopes

As soon as all sill-floor timber has been placed, the sets are lagged over with a double floor of 3 by 12-in. planks and stoping starts on the first floor at the original raise. The ground is blasted out around the raise as high as safety will permit, the ore is then removed and filling poured in from the level above, the waste taking its own angle of repose. When the waste filling is about 3 ft. from the back of the stope, it is roughly leveled off and floored with 3-in. by 12-in. by 5-ft. planks. Cleats of scrap timber are nailed to the floor to enable machinemen to move around easily and a cut about 6 ft. high is taken each side of the raise. When this cut is completed, the floor is taken up and piled out of the way and the opening is filled as before; the flooring is again laid and another cut taken. The flooring is used until it is worn out.

As the stope passes the temporary chute, the timber from this chute is salvaged for use elsewhere. As the toe of the incline reaches the permanent chute, the original raise timbers are salvaged as the stope progresses and are used to build up the permanent chutes and manway.

In all main drifts 2-in. air lines and ¾-in. water lines are carried, 1-in. air lines and ½-in. water lines are run down each original raise and up each center manway, so that drilling connections may be made at either the top or the bottom of the incline.

As stopes hole through to the level above, drift timbers are caught up and held in place until they can be supported by 8 by 8-in. sets placed upon the filling; these are eventually filled in and the original drift left intact.

Waste fill for stopes is obtained from development work, from shrinkage stopes above the ore zone, and from old filled stopes where no damage can be done by allowing them to cave.

Drilling and Blasting

All stoping is done with wet hand-rotated stopers using ⅞-in. quarter-octagon hollow steel. The starter bit is 1¾ in. and decreases ⅛ in. on

each length of steel. Thirty-five per cent. gelatin powder is used in blasting all holes in stopes.

A grab sample is taken from all rounds blasted and a copy of the assays is furnished to bosses daily. All broken material is hand trammed in 16-cu. ft. end-dump, roller-bearing cars run on 12-lb. rails.

The method adopted for drilling drifts, raises, and shaft sinking do not call for any special mention. This work is usually done on contract, the company providing all tools, equipment, and supplies; and the contractor providing all labor. The average price paid for drifting is $4.40 per linear foot with a minimum cross-section of 5 by 7 ft. The price for raises with a minimum cross-section of 5 by 10 ft. is $4.50 for the first 50 ft. and $5 per foot for the remainder. Shaft sinking averages about $50 per foot, depending largely on the character of rock being drilled and the amount of water likely to be encountered. All drifting and shaft sinking are done with water Leyners using 1¼-in. hollow round steel with double-taper cross bits. The starter bit is 2 in. and decreases ⅛ in. on each length of steel.

Blasting in all development work is done with 40 per cent. 1⅛-in. gelatin powder. All development is carefully sampled and accurate assay maps brought up to date every 30 days. About 1 ft. of development is done for every 12 tons of ore mined.

Costs

Tables 1 and 2 show average detailed costs of stoping and development work per ton of ore for 85,211 tons mined in 1923. These costs constitute approximately 50 per cent. of the total cost of mining. At

TABLE 1.—*Underground Mining Costs*

Stope	Labor	Explosives	Timber	Air	Hoist	Tram	Miscellaneous*	Total per Ton
812	$1.15	$0.21	$0.14	$0.48	$0.26	$0.14	$0.46	$2.84
901	1.29	0.15	0.13	0.61	0.26	0.14	0.42	3.00
902	1.67	0.31	0.27	0.48	0.26	0.14	0.42	3.55
904	1.55	0.24	0.40	0.41	0.26	0.14	0.42	3.42
1002	1.12	0.29	0.19	0.35	0.26	0.14	0.44	2.79
1004	0.88	0.37	0.14	0.43	0.26	0.14	0.42	2.70
1101	1.09	0.14	0.13	0.31	0.26	0.14	0.42	2.49
1102	1.32	0.24	0.25	0.37	0.26	0.14	0.42	3.00
1104	1.11	0.26	0.16	0.47	0.26	0.14	0.42	2.82
1202	0.96	0.19	0.27	0.26	0.26	0.14	0.42	2.50
1302	1.10	0.26	0.33	0.36	0.26	0.14	0.42	2.87
Average..	$1.20	$0.24	$0.22	$0.41	$0.26	$0.14	$0.42	$2.90

* Miscellaneous charge includes supervision, engineering, assaying, pipe, nails, carbide, track, steel, small supplies, etc.

present 30 miners break all ground in development work and stopes, including waste to fill stopes, and supply 300 tons of ore daily. There is an average of 100 men employed underground and a total of 135 at the mine. All labor in stopes is on a "day's pay" basis.

TABLE 2.—*Development Costs per Foot in 5 by 7 Ft. Drifts and Crosscuts and 5 by 10 Ft. Raises*

Place	Labor	Explosives	Timber	Air	Miscellaneous	Total per Foot
912 drift	$4.40	$2.13		$1.32	$1.04	$8.99
922 drift	4.40	1.39		1.05	0.58	7.42
926 drift	2.41	1.54		0.97	0.60	5.52
1006 drift	4.40	1.54		1.02	0.58	7.54
1202 drift	4.55	1.84		1.20	0.81	8.40
1402 drift	3.89	2.17		1.07	0.62	7.75
924 crosscut	4.18	1.51		1.08	0.40	7.17
903 crosscut	3.78	0.99		0.88	0.35	6.00
1404 crosscut	4.43	1.39		0.75	0.35	6.92
1406 crosscut	4.16	1.98		1.34	0.35	7.83
1006 raise No. 16	5.01	2.43	$1.69	2.87	0.38	12.38
1102 raise No. 8	4.83	2.00	1.88	2,37	0.47	11.55
1102 raise No. 9	4.50	2.10	1.87	1.90	0.45	10.82
1202 raise No. 4	4.75	2.25	1.86	2.53	0.51	11.90

TABLE 3.—*Wage Scale Effective March* 16, 1923

	Per Shift		Per Shift
Shaftmen	$6.05	Power-house engineman	$5.45
Miners	5.45	Pumpmen	5.75
Muckers	4.84	Pump repairmen	5.75
Timbermen	5.75	Blacksmiths	5.75
Cagers	5.45	Blacksmith's helper	4.54
Cagers' helper	4.84	Steel sharpener's	5.75
Top lander	4.84	Steel sharpener's helper	4.54
Pipemen	5.15	Machinist	5.75
Tool nippers	4.84	Machine repairmen	5.45
Scavanger	4.84	Power-house firemen	4.84
Trackmen	5.45	Chain gang	4.84
Crushermen	5.15	Timber framers	5.75
Hoisting enginemen, (second-motion double drum)	6.05	Watchmen	4.54
		Electricians	5.75
Hoisting enginemen, (second-motion single drum)	5.45	1st class surface labor	3.94
		2nd class surface labor	3.03

The labor turnover is about 15 per cent.

MACHINERY AND SURFACE PLANT

The surface equipment consists of an Allis Chalmers double-drum hoist driven by a 250-hp., 440-volt, 25-cycle electric motor, each drum holding 1800 ft. of 1⅛-in. 6 × 19 Lang lay cable. All hoisting is done in

counterbalance at a speed of 700 ft. per min. Ore is hoisted from the pocket through two compartments in 4-ton skips and dumped direct into a choke feed Austin No. 7½ gyratory crusher, which breaks to about 2 in. This material then passes through a trommel and the oversize is fed to a Symons 48-in. vertical disk crusher, the final product being ½ in. A belt conveyor carries it to the ore bins, and from there it is taken to the mill, ½ mile away, by a Westinghouse 6-ton locomotive operating over a 24-in. gage electric railroad on 550-volt d.c. current.

A cage for hoisting men is hung under the skip, and provision is made for connecting a second cage underneath if necessary. An unbalanced dinkey cage is operated in the third compartment by a 10 by 18-in. duplex, direct-acting single-reel Ottumwa hoist; this cage is used only to lower supplies. Skips and cages are both equipped with safety catches and are inspected daily.

Pumping

At present, the water is handled by electrically driven pumps in two lifts, from the 1500 to the 1300-ft. level, and from this level to the mill on the surface. Both pumps are on the 1300-ft. level. A Lane & Bowler 500-gal. deep-well pump, six-stage, driven by a 40-hp. motor lifts the water from the 1500-ft. level through an 8-in. pipe and discharges into two 60,000-gal. concrete sumps on the 1300-ft. level. The water gravitates into a 300-gal. Aldrich quintuplex plunger pump driven by a 150-hp. motor and pumps through a 6-in. pipe direct to the mill on the surface, about ½ mile from the collar of the shaft.

Air Compression

Air for drilling is furnished by a steam-driven 3000-cu. ft. O. R. C. Ingersoll-Rand compressor. A Sullivan 1500-cu. ft. tandem, compound, direct-connected, steam-driven compressor is idle at present but can be used in an emergency.

Ventilation

Ventilation is provided by one Sturtevant multivane fan pulling 65,000 cu. ft. at 3½-in. water-gage pressure, driven by a 75-hp. motor, belt-connected. This exhaust fan is at the collar of the Williams shaft and is operated 14 hr. per day. Air is drawn down through the Iron Cap shaft to the lowest mine level and allowed to work upward through the stopes, finally finding its way out through the Williams workings and up the Williams shaft.

Lighting and Signaling

Stations are lighted by 32-c.p., 100-watt, 110-volt electric bulbs; main tramming drifts are lighted by 16-c.p. 40-watt, 110-volt electric bulbs. Lights in stopes are provided by carbide lamps carried by each miner.

Western Electric, local-battery system telephones are located on each shaft station and at the collar of the shaft. All signals between cagers and hoisting engineer are over an electric signal system, supplemented by rope bell.

All electric power used is bought from the Inspiration Copper Co. at Miami, Ariz., and brought over a private line a distance of 7 mi. All steam power is furnished by three 250-hp. Babcock & Wilcox water-tube boilers.

DISCUSSION

A Member.—What do they put on the filling?

A. L. Walker.—They put lagging and use that lagging until it is worn out, gradually moving it from one place to another.

A. Neustaedter, Roselle Park, N. J.—Do they put up square-set raises?

A. L. Walker.—They put up square-set raises, but use square sets only when it is imperative. If the ground is at all soft or dangerous they use stulling.

A. Neustaedter.—Cribbing would not do.

A. L. Walker.—It might in certain cases. Of course, in the old days, the square-set system was used altogether. In the Old Dominion mine, where the orebodies were 40 or 60 ft. wide and sometimes 60 ft. high, square sets were used altogether.

A. Neustaedter.—Do they work the square-set stopes on the rail too?

A. L. Walker.—Yes; all the orebodies above the eighth level in the Old Dominion were worked with the square-set system. About 1893, however, that system became so expensive that we developed a system of heavy stulling to support the roof whenever possible.

Mining Methods of Verde District, Arizona

By C. E. Mills*, Jerome, Ariz.

(New York Meeting, February, 1923)

The Verde mining district is in Yavapai County, in north-central Arizona. Jerome, the principal town, has a population of 6000 and the two important mines of the district—the United Verde and the United Verde Extension—lie almost within its limits. Perched high on the southwest side of the Verde valley, at elevations of 4800 and 5500 ft. (1463 and 1676 m.) respectively, both mines have long tunnels from their lower levels through which the ore is hauled to the smelters.

Clarkdale, the smelter town for the United Verde mines, lies in the Verde valley 4.1 miles (6.5 km.) distant in an air line from Jerome and at an elevation of 3560 ft. The two towns are connected by the standard-gage railroad of the Verde Tunnel and Smelter R. R., a subsidiary of the United Verde Copper Co. This road is 11 miles long and has a compensated grade of 4 per cent. At Clarkdale it connects with the Verde valley branch of the Santa Fe railroad.

Clemenceau, the smelter town for the United Verde Extension mine, lies 3.5 miles (5.8 km.) in a southeasterly direction from Clarkdale, at an elevation of 3420 ft. It is connected to the Verde Valley branch of the Santa Fe railroad at Clarkdale by the Arizona Extension R. R., a subsidiary of the United Verde Extension Mining Co. The portal of the haulage tunnel, at an elevation of 3757.5 ft., is also connected with the smelter town by 4 miles of standard-gage track belonging to the same company.

In the vicinity of Jerome, the great southern escarpment of the Arizona plateau forms the northeast side of the Verde valley. The general level of this escarpment, or "rim" as it is called, is about 4000 ft. above the Verde River. A corresponding elevation is reached only by the highest peaks of the Black Hills, which bound the valley to the southwest. Two lava-capped mesas, known as "Mingus" and "Woodchute mountains," form the culmination of the Black Hills. That part of the Old Black Hills mining district which lies between their crests and the Verde River is now known as the Jerome, or Verde, mining district. If the smelter towns are not included, a strip about 7 miles long by 3 miles (11 by 5 km.) wide, embraces practically all of the properties belonging to the Jerome district. This strip lies parallel to and along the side of the Verde valley from First View, 2 miles northwest from Jerome, to the

*Chief Mining Engr., United Verde Copper Co.

vicinity of Allen Springs, at the south end of Mingus mountain and all within the boundaries of the Prescott national forest reserve.

History

The Verde mining district dates back to 1877, when Al Sieber, John Dougherty, and Capt. J. D. Boyd located three claims on what is now the property of the United Verde Copper Co. No work of importance was done until 1883, when a company was organized with James A. McDonald as president and Eugene Jerome as secretary. A 50-ton furnace was put in operation and a phenomenal run made on oxidized ores high in silver and gold. In 1884, operations were suspended because of difficulty in bringing in supplies, all material having to be hauled from Ash Fork. In 1887, Governor Trittle of Arizona secured a lease on the property but found operating conditions unprofitable. In 1888, Senator W. A. Clark, of Montana, leased the property and purchased control in 1889. Until 1894, when the United Verde and Pacific narrow-gage railroad was completed, connecting Jerome Junction, a point on the Santa Fe, with Jerome, all supplies were hauled from Granite, 28 miles distant, at a cost of $9 per ton. From 1894, the United Verde mine has gradually developed into one of the largest copper mines in the world.

The other large property of the district, the United Verde Extension, was first located by J. J. Fisher, a mineral surveyor, with whom was associated L. E. Whicher, of New York, a broker and promoter. The property was worked intermittently until the death of Fisher, in 1911. Major A. J. Pickeral, of Prescott, who had assisted Mr. Fisher financially, interested James S. Douglas and the Douglas-Tenor syndicate took control and started to sink the second shaft, the Edith. Results were discouraging until December, 1914, when a small high-grade stringer of chalcocite was encountered on the 1200-ft. level. In January, 1916, the present high-grade body of ore was discovered on the 1400-ft. level and was soon found to be one of the largest high-grade sulfide deposits of the world. This strike at the United Verde Extension attracted attention to the district and many new companies were organized during the succeeding two years.

From 1894, the production of the district has steadily increased until it now ranks as the third largest producer in the state, exceeded only by the Globe-Miami and Bisbee districts. Its production places it sixth among the copper camps of the United States.

The two mines furnish practically the entire production of the Jerome district. During 1920, the United Verde Extension averaged 3,500,000 lb. of copper per month and the United Verde 6,000,000 lb. In this year, these two mines yielded 8.74 per cent. of the total copper production of the United States and 19.1 per cent. of the copper production of Arizona. The recovery of the precious metal content is also

important, the district being credited with 1,300,000 oz. of silver and 22,600 oz. of gold for the year 1920.

The usual and maximum size lode claim is 600 by 1500 ft. (182.9 by 457.2 m.) according to Sec. 2320 of the Revised Statutes of United States Mining Law and Regulations thereunder. Possibly 25 per cent. of the claims of the Verde district are patented and held in fee, the greater portion consisting of unpatented ground.

Geology of Jerome District

Pre-Cambrian

The oldest rocks exposed in the Jerome district are of pre-Cambrian age. They comprise a greenstone complex, which is usually siliceous in composition, yet contains enough chloritic material to give it a green color when freshly broken and an iron-stained appearance when weathered. This complex indicates a period of important volcanic activity. The term "greenstone" is a non-committal one and covers much material, the exact nature of which is difficult to determine. It can be recognized along the main fault where the pre-Cambrian is exposed, from the Copper Chief mine in the southern end of the district north to the outskirts of Jerome.

Younger than the greenstone is a series of clearly bedded sediments which, although containing much fragmental material, are evidently water laid and in part well sorted. This series records the gradual erosion and slow sedimentation following the period of volcanic activity. The bedded sediments are closely folded and the dips of the beds prevailingly steep. A great variety of material is included, from fine-grained quartzite and slates to thick bedded conglomerates.

The deposition of these sediments was followed by a period of deformation, probably related to the approaching invasion of a granitic batholith, which squeezed the bedded sediments into close folds trending approximately N 20° W. The greenstones in the southern end of the district were folded along lines trending nearly east and west.

Following this deformation, the region was invaded by masses of quartz porphyry, presumably an outlying phase of the Bradshaw granite, which underlies the greater part of Yavapai County to the southwest of Jerome. The quartz porphyry is usually light in color with quartz and feldspar phenocrysts varying in abundance and the ground mass varying in fineness. The usual composition is that of a normal rhyolite. Mine workings show that the quartz porphyry predominates throughout a large area in the northeastern part of the district. Although usually very massive, later deformation has rendered certain portions schistose; the most important schistose area being that of Cleopatra Hill and the United Verde mine.

After this last deformation, masses of augite-diorite were intruded into the quartz porphyry and older rocks in the vicinity of Jerome. The

principal mass of this diorite is ¾ miles long and ¼ mile wide (1.2 by 0.4 km.). It trends northeast and southwest and forms the hanging wall of the United Verde ore zone. Farther north another mass of this diorite is partly exposed. It also shows in the Jerome Verde workings to the southeast, and in the United Verde Extension workings just east of the fault. Practically no evidence of contact metamorphism has been observed along either the quartz porphyry or the diorite contacts.

After the intrusion of diorite occurred, the ore deposits, which will be discussed later, were formed. Following the formation of the large schist replacement orebodies, but before the mineralization was completed, a series of small diorite or andesite dikes cut the ore masses, diorite and other pre-Cambrian formations. These dikes are locally called "water courses" and vary in thickness from a few inches to 40 ft. but the greater number are from 1 to 4 ft. (0.3 to 1.2 m.) thick. In the vicinity of Jerome, they have a general east-west trend, cutting the locally schistose quartz porphyry nearly at right angles.

The quartz veins of the southern part of the district were probably formed after the principal mineralization and may be related to the same fissuring as the dikes.

The intrusion of the diorite marked the end of the intense deformation that affected the older rocks. The period of deformation and igneous intrusion preceding the deposition of orebodies undoubtedly produced a rugged and mountainous land high above the sea. The erosion period, which reduced the region to practically a flat surface, was no doubt long and during it the orebodies were exposed, portions worn away and other portions enriched by secondary chalcocite.

Paleozoic

The district was probably submerged in middle Cambrian time. A thin layer of sand and pebbles varying from nothing to 100 ft. in thickness formed the basal sandstone and filled the minor irregularities in the pre-Cambrian surface. More or less clayey material was deposited above the sandstone and is now represented by 10 to 20 ft. (3 to 6 m.) of red or greenish yellow shale.

Overlying the basal sandstone are from 300 to 500 ft. (91 to 152 m.) of limestone, all or in part of Devonian age, from 300 to 500 ft. of limestone of Mississippian or lower Carboniferous age and from nothing to 500 ft. of red sandstone and shale of the Permian age. The formation of the deposits of each of these four periods was preceded and followed by periods of uplift and erosion.

Cenozoic

The Tertiary record is one of volcanic activity, terrestrial deposition, and faulting. The Paleozoic formations were trenched with steep-sided

gulches, and the Verde valley seems to have existed somewhat along its present lines. Many of the gulches were more or less filled with poorly stratified gravels and boulder material before being covered with lava. The volcanic field of the San Francisco Mountains is not far north of the Jerome district and the great outpouring of basaltic lavas, which were fluid and spread out for long distances, covered a large part of northern Arizona. The hills of Jerome are still capped with about 700 ft. (213 m.) of this lava. The lava flows of late Tertiary time dammed the Verde valley and formed a lake in which was deposited the impure marly limestone which covers much of the present valley.

Following the outpourings of basaltic lava, came a period of normal faulting which uplifted the west side of the Verde valley more than 2000 ft. (609) m.), possibly as much as 5000 ft. The trend of the principal breaks varies from north and south to northwest and southeast.

The interrupted drainage of the lake period and relief produced by faulting promoted the formation of extensive deposits of poorly assorted gravels and boulders differing from those underlying the lava chiefly in the presence of fragments and boulders of basalt.

The uplifted scarp of one of the principal faults was especially subject to erosion, so that a strip of pre-Cambrian has been exposed along the southwest side of the Verde valley, trending in a direction approximately S 37° E from Jerome. This fault, known as the Main or Verde fault, passes through Jerome and practically divides the district in two parts. East of this fault, the pre-Cambrian formations are deeply buried.

Ore Deposits

United Verde Mine

The United Verde mine lies west of the Main fault in the area of exposed pre-Cambrian rocks and on the west side of Jerome. The ore zone is approximately 500 by 1100 ft. (152 by 335 m.) in cross-section. The orebodies are of the schist replacement type and the mineralization was aided by the concave margin of the United Verde diorite, which formed a steeply pitching inverted trough and localized the solutions in their upward course. The concave portion of the diorite enclosed portions of the bedded sediments, which were altered to black schist, and a portion of the large schistose area of quartz porphyry. Thus the diorite contact and the permeable quartz-porphyry sericite-schist favored the penetration of the solutions, while the less permeable black schist was most favorable to replacement. Some quartz porphyry and some greenstone schist were replaced but the diorite was not affected to any appreciable extent by the mineralizing solutions.

Quartz, pyrite, chalcopyrite, sphalerite, specularite, dolomite, calcite, bornite, tetrahedrite, and galena belong to the primary mineralization.

The main bodies developed are nearly massive sulfides, in which the pyrite greatly predominates. Chalcopyrite is the most important and, next to pyrite, is the most abundant of the primary sulfides. Where this occurs in sufficient quantity, the sulfides are mined as copper ore. Less than one-fifth of the massive sulfide area is of commercial grade. The shape of these richer portions is roughly lenticular and is largely influenced by the structure of the replaced schist. The primary sulfides average from 1 to 2 oz. of silver and from 0.02 to 0.04 oz. of gold. The highest silver values occur with sphalerite and quartz. Quartz occurs in the peripheral area of the pyrite and large masses of relatively pure quartz occur between the massive sulfide and the diorite hanging wall. This is a characteristic jaspery quartz which occurs in many parts of the district.

The oxide zone extends down to the 160-ft. level. Oxidation of the sulfides with little copper enrichment preceded the actual erosion of the surface and the enrichment of the pre-Cambrian erosion period has been in a large measure destroyed, instead of increased. Some chalcocite remains in the upper levels of the United Verde mine, in localities now inaccessible. Since the uncovering of the orebodies by post fault erosion, conditions apparently have not been favorable to secondary sulfide enrichment. The small gold and silver values of the primary sulfides are concentrated sufficiently in the oxide material so that much of it is of commercial grade. It carries about 1 per cent. of copper, largely in the form of finely divided cuprite.

United Verde Extension

The United Verde Extension orebody lies to the east of the Main fault. The oxide and chalcocite zones of the pre-Cambrian period of erosion and enrichment are still protected by approximately 750 ft. of sedimentaries and lava flows.

The main sulfide mass of the United Verde Extension ore deposit has an irregular pipelike form, similar to that of the United Verde. The mass is entirely surrounded by quartz porphyry and seems to have been localized by a fissure or shear zone nearly at right angles to the normal trend of schistosity of the schistose porphyry. It is a replacement of the schistose porphyry and of lesser amounts of greenstone schist.

The orebody was a pyrite-chalcopyrite replacement similar to that of the United Verde and was probably formed at the same time, by the same agencies. The ore deposit was undoubtedly exposed to the erosion period preceding the deposition of Cambrian sandstone and the pyrite-chalcopyrite largely changed to chalcocite by secondary enrichment. The oxide zone, or mass of gossan material, is about 450 ft. thick and the secondary chalcocite zone about 400 ft. The original pyrite-chalcopyrite zone below the secondary zone has not been explored to any extent.

The main orebody consists of an irregular lens of chalcocite and pyrite, oval in shape, averaging 400 ft. long by 220 ft. wide (122 by 69 m.) and extending vertically upward to a point half way between the 1300- and 1200-ft. levels. The ore varies from pure masses of chalcocite to material in which pyrite predominates. It is not uncommon for large areas to average from 30 to 40 per cent. copper.

Copper Chief-Iron King Deposit

The Copper Chief-Iron King ore deposit is similar to, but is much smaller than, the United Verde. It is 4 miles south on the upper side of the Main fault. The original sulfide mass was a replacement of schistose portions of the fine-grained margin of a granitic porphyry mass, which intruded siliceous and slightly schistose greenstones. The orebody is important chiefly because of the gold-silver values in the oxidized gossan material. The sulfide mass has been shown to pinch out on the lower workings of the Copper Chief and Iron King mines.

Exploration

On the upper side of the Main fault where the ore-bearing formations are exposed, prospect work is largely guided by surface conditions. On the lower side, where the ore-bearing rocks are covered with sedimentaries and laval flows, underground exploration is sometimes justified. The fact that the United Verde Extension opened up a high-grade orebody below the fault, with no surface indications, has largely been responsible for the organization of new companies since 1916.

In exploring the pre-Cambrian formations beneath the Paleozoic cover, the character of the rocks is the chief factor in guiding the work. Large masses of well-developed schist are essential to the occurrence of important orebodies of the schist replacement type. The nearly black, chloritic schist and light-colored sericitic, quartz-porphyry schist are the most favorable varieties. Contacts between the schist and a relatively massive and impervious formation are favorable localities, especially where the impervious material forms the hanging wall to the schist.

Iron oxide weathered from the highly ferruginous Cambrian sandstone and deposited in the underlying formation has sometimes been mistaken for gossan material. Similarly, the iron-staining developed by the weathering of the more chloritic rocks of the Jerome district has been incorrectly referred to as evidence of mineralization.

Because exploratory work can best be accomplished by crosscutting the steeply dipping geologic boundaries and structures in the pre-Cambrian, diamond drilling from the surface has not proved very satisfactory. Most of the drilling in the district is conducted from mine workings to determine favorable geological formations, as well as to locate orebodies.

United Verde Mine

The geology and structural conditions of the United Verde orebody have been described. The structural conditions are more clearly shown by the geological plan of a typical mine level, Fig. 1. The ore-

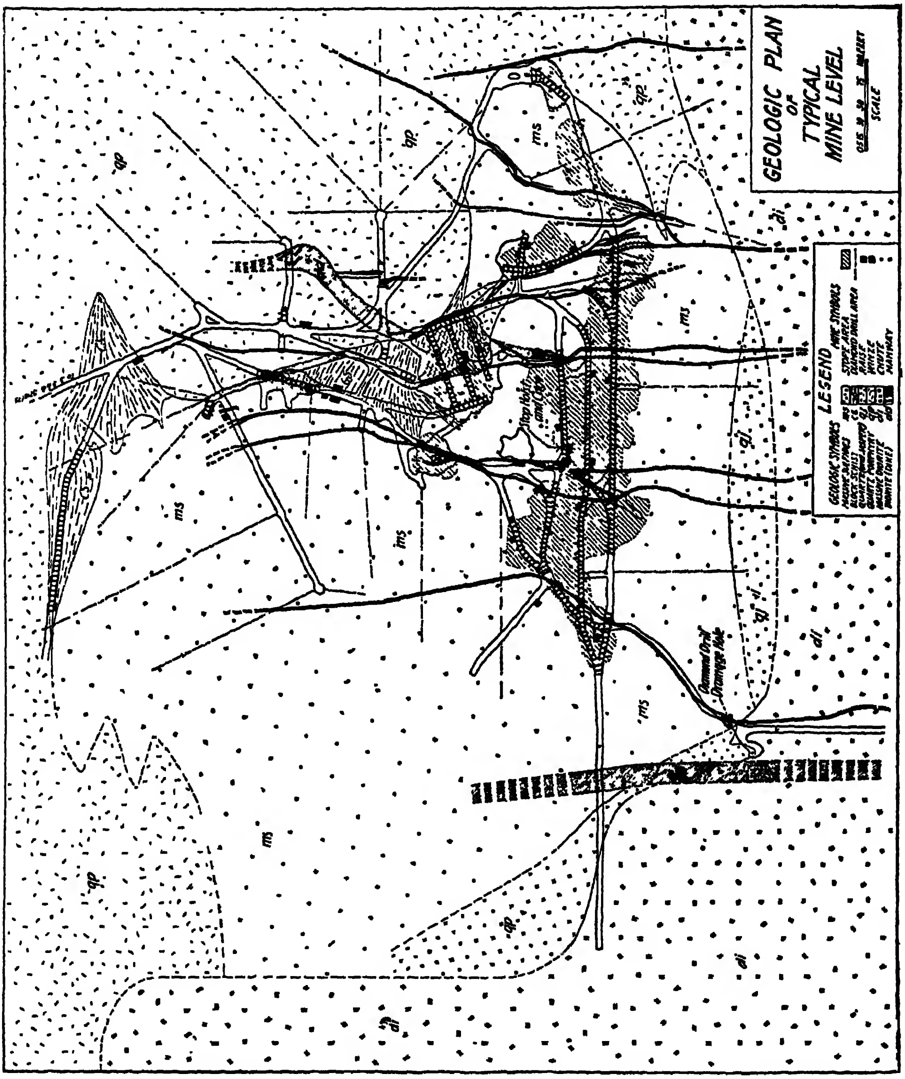


Fig. 1.—Geologic plan of typical mine level.

bodies are steeply dipping lenses, varying in section from a few feet to 200 ft. wide by 500 ft. long. (60 by 152 m.). As before stated, secondary enrichment of the sulfide mass is of little importance in the orebodies exposed at the United Verde mine.

Classes of Ore

For the convenience of smelter operation, the ores mined are divided into four classes. Each class must be handled separately in all mining operations.

Oxide ore is mined from the oxidized zone above the 160-ft. (48-m.) level. It contains a small amount of altered primary sulfides but in general has a low copper content. An average assay shows the precious metal content as 0.2 oz. gold and 8.0 oz. silver. This class of ore is now handled entirely by steam-shovel operations.

Iron ore is mined from the stopes within the massive sulfide areas of the mine. If the silica content is 15 per cent. or more, it is classed as silica ore and is diverted to the silica ore bins.

The ore from all black-schist stopes and from the quartz-porphyry stopes assaying less than 50 per cent. free silica is also classed as silica ore. The analysis given in Table 1 shows that the term "silica ore" is somewhat of a misnomer, as the iron content is often in excess of the silica contained.

TABLE 1.—*Tonnage and Analyses of Ores*

Class of Ore	Dry Tons	Total Shipments, Per Cent.	Copper, Per Cent.	Gold, Ounces	Silver, Ounces	Iron, Per Cent.	Insoluble, Per Cent.	SiO_2, Per Cent.	Al_2O_3, Per Cent.	Sulfur, Per Cent.	Zinc, Per Cent.
Oxide..........			1.42	0.225	8.37	31.5	40.4	34.1	5.7	4.2	0.2
Iron...........	81,707	52.7	7.0	0.035	1.46	31.6	14.9	12.2	2.1	36.1	3.5
Silica..........	57,900	37.3	4.6	0.019	0.98	19.6	39.0	35.1	8.8	16.5	1.8
Converter......	15,355	9.9	1.0	0.058	1.64	6.0	77.1	73.2	6.0	3.5	0.5
Precipitates.....	100	0.1	81.79			4.3	1.8	1.3	2.9	1.0	0.6
Totals.......	155,062	100.0	5.56	0.031	1.30	2.46	30.1	26.9	5.1	25.5	3.0

NOTE.—This includes 1599 tons of silica ore and 11,657 tons of converter furnished by shovel operations.

An ore high in free silica is essential for use as flux in the converters and for fettling the reverberatory furnaces. A small tonnage of ore meeting this requirement is mined from stopes in quartz porphyry near contacts with the sedimentary schists. A second possible source of supply is a low-grade, secondary enrichment orebody in quartz porphyry, some distance from the main orebodies, extending from the 160-ft. level. The large bodies of jaspery quartz associated with the sulfide mass often carry gold and silver values, and constitute an important source of "converter ore." This class of ore, at present, is almost entirely supplied by the steam-shovel operations.

In addition to these classes of ore, a small amount of precipitate is recovered from the copper-bearing mine waters and shipped directly

to the smelter. Experimental work is also being conducted by Joseph Irving, formerly in charge of the leaching experiments at the Copper Queen Branch, Phelps Dodge Corpn., on a 5000-ton low-grade sulfide dump, to determine the possibility of recovery of copper by heap leaching, using acid mine water after it passes over the precipitating troughs as the solvent.

The average analysis and tonnage of each class of ore shipped during June, July and August, 1922, are given in Table 1 and their approximate mineralogical composition in Table 2.

TABLE 2

	Ores, Per Cent. of			
	Oxide	Iron	Silica	Converter
Pyrite	5	59	28	5.75
Chalcopyrite		18	16	1.00
Sphalerite		4	2	0.50
Chalcocite	2			1.50
Cuprite, native copper and carbonates				0.25
Cuprite	1			
Sulfates and carbonates of FeCu and lime	3			
Iron oxides	46	2	2	11.00
Ferruginous chlorite			38	6.00
Quartz	28	5	5	54.00
Silicates (sericite, hornblende, kaolin)	15	10	9	20.00
Calcite and siderite		2		

STOPING AREAS

The total area of stoping ground developed to date on the sill floors of four representative lower levels is given in Table 3. These areas do not include material less than 2.5 per cent. copper. To maintain the ore reserves at the present figure and produce 80,000,000 lb. of copper per year, it will be necessary to average about 100 ft. of development in depth each year.

TABLE 3.—*Stoping Areas*

Stoping Area, Square Feet	Average Grade, Per Cent., Copper	Tons of Ore, per Foot of Depth	Tons of Copper, per Foot of Depth
38,400	5.22	4,266	222.7
46,700	8.01	5,495	440.1
102,760	8.01	12,089	968.3
58,280	6.77	6,856	464.2
Average..... 61,535	7.28	7,176	523.8

Conditions Determining Stoping Methods

The general conditions that govern the selection of a mining method are the dip, shape and size of the orebody, character of the walls, costs of material and character of the ore itself. Other important considerations are: the necessity for safe and efficient working conditions for the men, fire hazard, complete extraction of the ore, flexibility for stope development, ease of sorting waste, and possibility of future mining of low-grade mineralized areas on the borders of the present stopes. The stoping methods and their relative importance are given in Table 4, and descriptions of their application to the United Verde deposits are given later (page 402).

Table 4.—*Stoping Tonnage, June, July, August,* 1922

Method of Stoping	Dry Tonnage Mines	Per Cent. of Total
Horizontal cut and fill	92,925	66.3
Incline cut and fill	4,489	3.2
Square set and fill	8,455	6.0
Shrinkage and fill	2,702	1.9
Glory hole	31,677	22.6
Total	140,248	100.00

On account of the relatively high copper content of the ore and the fact that it is smelted direct, it is necessary to use a method of stoping that gives practically complete extraction and little dilution by waste. This condition makes it impossible to use any caving system and the character and concentration of the orebodies also eliminates the possibility of using top-slicing methods.

The incline cut-and-fill is seldom used because of the tendency of the ore to shatter well ahead of the ground blasted and the consequent danger of working under the brow. The massive sulfides break in large pieces, which will stand on a slope of 60° and which entail much labor in blockholing and breaking with hammers. The saving effected in mucking labor in an inclined stope and the more pronounced saving in handling waste fill are practically offset by a higher cost of handling timber and steel and the difficulty of erecting bulkheads on an inclined slope, as compared with the horizontal cut-and-fill method. This method is occasionally used in small narrow stopes where the walls are not strong enough to permit shrinkage. It was also used in one case of removing a floor pillar where the shrinkage stope above had not been filled.

Shrinkage stopes can only be used in hard ground in massive sulfide areas. This method shows a lower cost for timbering and waste fill than any other and permits a large reserve of broken ore. As the sulfide

ore has a tendency to break in large pieces, the cost of drawing ore from chutes on the levels is greater than where grizzlies are used in the stopes. With this method, it is also difficult to sort waste. The character of the walls, size of the orebodies and occurrence of occasional slips and water courses limits this method to a few stopes.

The horizontal cut-and-fill stope overcomes most of the objections to the preceding methods. It is flexible, in that horses of waste can be left unmined or the stope offset on any floor where the values carry into the wall. It is not uncommon to leave waste pillars, but actual waste broken is limited to small diorite or andesite intrusions varying from a few inches to several feet thick and probably does not exceed 0.5 per cent. of the tonnage mined. The walls are easily prospected by crosscuts, which is particularly essential in schist stopes. This method also facilitates the sorting of waste, erection of bulkheads, handling of timber and supplies, and permits the ore to be broken or bulldozed on grizzlies rather than at the chutes on the level. Because of these advantages over other methods, the horizontal cut-and-fill is generally used.

Where the ground is not strong enough for the horizontal cut-and-fill method, the square-set method is used. This is especially true in the black-schist stopes and occasionally in the highly mineralized sulfide stopes, where the ground is so badly broken. It is also necessary to use square sets in removing floor pillars where the stope above has been mined and filled with waste.

Until the introduction of the cut-and-fill method under the superintendence of Robert E. Tally, practically all stoping was done by square-set and fill methods. Square-set stoping is adapted to the United Verde orebodies, except for the extreme fire hazard involved and the high cost of labor and timber. Compared with the cut-and-fill system, a square-set stope yields a much smaller daily tonnage, is more difficult to supervise and ventilate and will not permit the accumulation of a reserve of broken ore. Therefore it is used only where the ground is not strong enough for the cut-and-fill method.

Underground Development Plans

The general arrangement of drifts, raises and crosscuts is shown in Fig. 1. All exploration and development work, stoping methods, tramming and hoisting are planned with the definite purpose of providing and maintaining an underground production of 3000 tons of ore per day. To do this, it is necessary to develop 100 ft. in depth each year and drive 25,000 ft. of drifts and raises for exploration and development. During the last few years, an average of 34 tons of ore has been developed per foot of development work. This high figure is due to the character of the orebodies, which are large, persistent and concentrated within a relatively small area.

Level Interval

The level interval above the 1000-ft. level is 1000 ft. This interval is satisfactory for square-set stopes, but with the cut-and-fill method, the last three or four floors, or practically 20 per cent. of the total stope tonnage, if under a filled stope on the level above, must be mined by square sets.

A 200-ft. interval (1000-ft. to 1200-ft. levels) proved unsatisfactory because of high maintenance cost of cribbed chutes through the stope fill, increased cost of raise developments and the necessity of sublevel work, half way between levels, for development purposes. The 150-ft. interval largely overcomes these objections and has been adopted on all levels below the 1200-ft.

General Development

In opening up a new level, the position of the ore is fairly well determined by a projection from the levels above. Knowing the position of the two main shafts and the general location of the orebody, or bodies, a fairly well-defined plan of development can be pursued. The main drift is run to intersect the center of the orebody, which is then developed, as described under "stoping methods." All new orebodies are developed by drifts and crosscuts, or by diamond drilling, followed by drifts and crosscuts.

Diamond Drilling

Diamond drilling at the United Verde mine serves two purposes: Long holes are drilled to locate and explore the different geological formations, while massive sulfide areas are developed and blocked out by numerous short holes. It has been found that the diamond-drill cores from schist areas cannot be relied on for accurate data on the mineralization, consequently, the drill is used to locate and determine the size of schist areas and the actual development is performed by drifts and crosscuts. The assays of cores from the massive sulfide areas have been found to check closely with subsequent drift development work. For this reason, and because of the high cost of drifting in such areas, exploration of the massive sulfide areas and establishing the boundaries of large sulfide orebodies is usually done by diamond drilling.

The cost of diamond drilling varies with the class and character of formation, length and angle at which the holes are drilled, and the diameter of the core removed. The ES bit, which gives a core $\frac{7}{8}$ in. (2.2 m.) in diameter, is used for most work. On holes over 1000 ft. in length, the E bit ($1\frac{5}{16}$-in. core) is used for the first 700 ft. and the ES bit for the remainder of the hole.

During the first six months of 1922, the diamond-drill footage was 3533 ft. total, or a monthly average of 589 ft. With the drill-run or bit-

setter at $5.25 per shift, helpers $4.25 per shift, and first quality carbons $77 per carat, the average cost per foot was as follows:

	Cost per Foot
Labor	$1.32
Supplies	0.01
Carbons	1.09
Power	0.43
Repairs	0.32
Total	$3.17

Drifts and Crosscuts

All main drifts and crosscuts have the same cross-section, 6 ft. wide by 8 ft. high (1.8 by 2.4 m.) in the rock section and 5 ft. by 7 ft. (1.5 by 2.1 m.) in the clear where timbered. At present 18-cu. ft. cars are used for both hand tramming and motor haulage, but the contemplated use of 30 or 40-cu. ft. cars on all new levels will require a larger size drift on motor-haulage levels.

Gangways under stopes are driven 7 ft. wide by 8 ft. high and a second cut 7 by 7 ft. is stoped from the back of the drift to make room for the gangway sets. Fig. 10 shows the typical cross-section and longitudinal section of a gangway drift, giving details of timbering above the set for the support of the filling. All gangways are 5 ft. by 9 ft. 10 in. in the clear.

Raises

The standard raise has a cross-section of 6 by 11 ft. (1.8 by 3.3. m.). If the ground is heavy, double cribbing, as shown in Fig. 8, is used, except that the outside of the cribbing is not lagged. Usually single cribbing 4 by 6 in. by 4 ft. is used. It is carried up as a manway compartment and is removed on completion of the raise, either to be used again or cut up for track ties.

All drifts have a grade of 0.4 per cent. in favor of the load; 18-in. gage is used on all levels with 40-lb. rails for motor-haulage levels and 16-lb. on the hand-tramming levels. The minimum radius used is 50 feet.

Mine Openings, Shafts, or Tunnels

The general arrangement of shafts and tunnels at the United Verde mine is shown by Fig. 2. Two shafts serve the mine, No. 5 handling all ore from below the 1000-ft. level, and No. 6 all men and supplies. There are two main tunnels. All supplies are handled from the 500-ft. level surface plant through the 500-ft. level tunnel to No. 6 shaft, whereas all ore is handled through the Hopewell haulage tunnel on the 1000-ft. level.

Prior to September, 1918, all men, timber and supplies were handled through shafts Nos. 3 and 4. No. 3 also hoisted all ore mined below the

1000-ft. level to the 900-ft. level for transfer to the loading bins of the Hopewell tunnel.

A new installation was necessary because of difficulty in handling the tonnage from the lower levels on double-decked cages and 18-cu. ft. cars and at the same time handling material and supplies. No. 4 shaft was only down to the 1000-ft. level, the upper part of No. 3 shaft was in moving ground and the collar was within the limits of the proposed steam-shovel pit. A new shaft, No. 5, was therefore planned to provide increased ore capacity and No. 6 shaft was later planned to handle men,

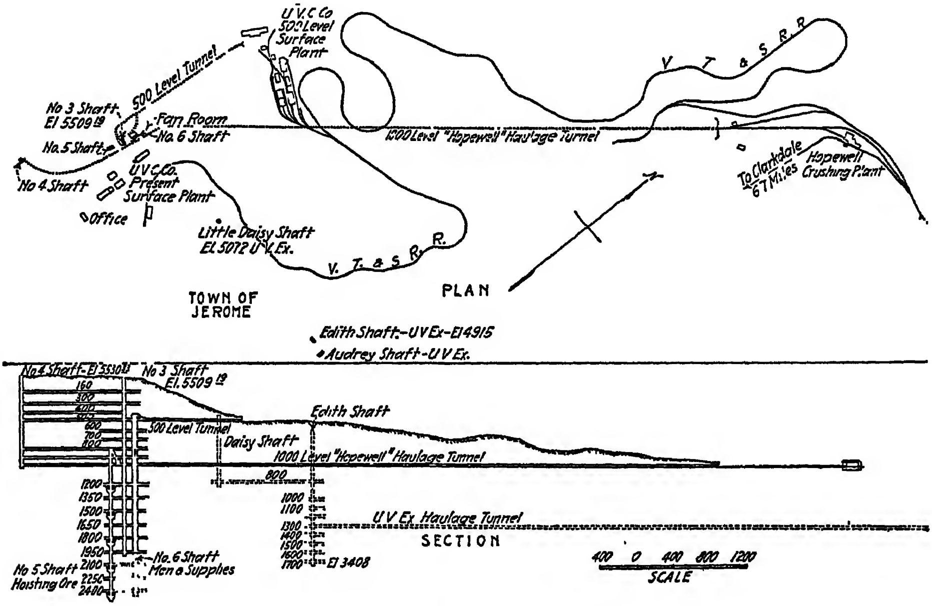


FIG. 2.—LOCATION OF SHAFTS AND TUNNELS, UNITED VERDE MINE.

timber, and supplies. The location of these new shafts was largely determined by the location of the main Hopewell haulage tunnel and loading bins.

No. 5 Shaft for Handling Ore

No. 5 shaft, Fig. 11, is 5 by 16 ft. (1.5 by 4.8 m.) in the clear and has three compartments. The hoisting compartments are 5 by 5 ft. and the manway is 4 ft. 4 in. by 5 ft. It extends from the 800-ft. level to the 2500-ft. level (518 m.) and is of reinforced concrete with a minimum wall thickness of 12 in. and 10-in. curtain walls.

No. 5 hoist room is located on the 1000-ft. level. It is 47 by 81 ft. in section by 22 ft. in height (14.3 by 24.6 by 6.7 m.) and is lined with reinforced concrete. A 20-ton crane facilitates the rapid handling of hoist-room equipment for repairs.

The level interval below the 1200-ft. (365 m.) level is 150 ft. (45 m.) and trolley motor haulage is provided on every other level. The skip pockets, therefore, are located below the 1200-ft., 1800-ft,, 2100 ft., and 2400-ft. levels, the two last named being in course of construction.

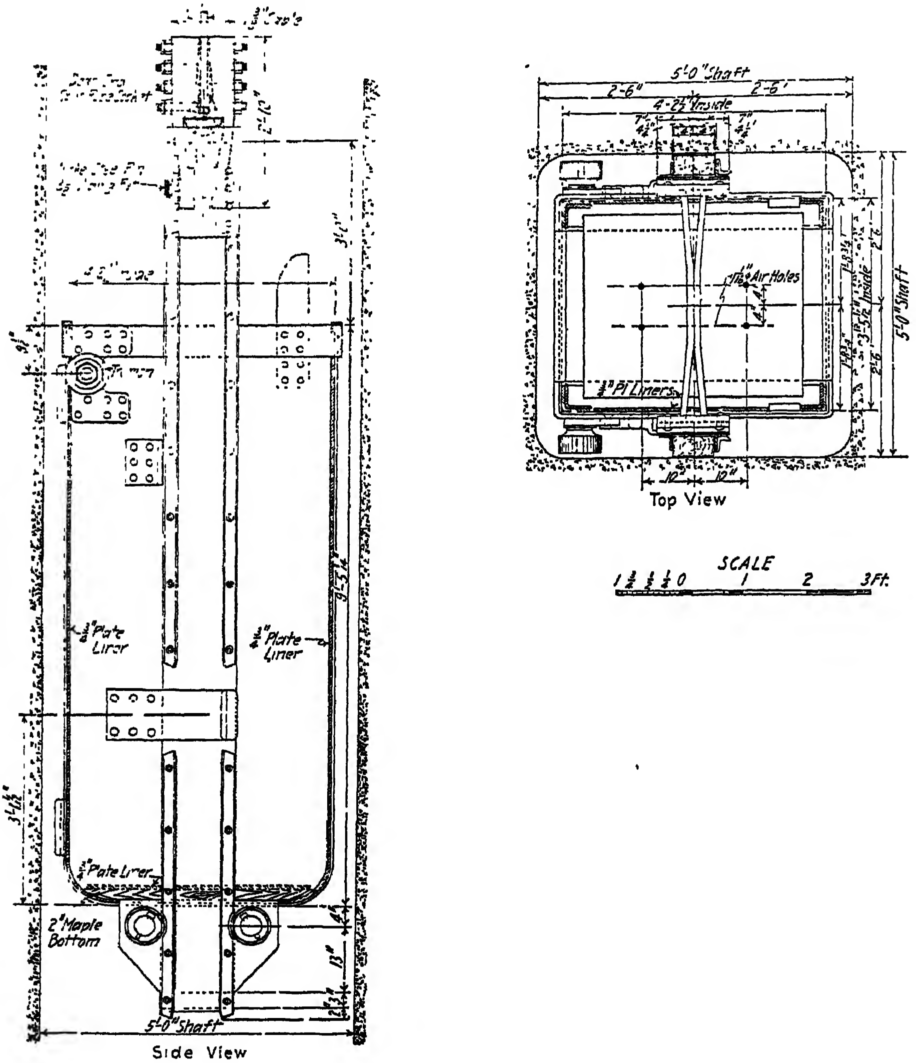


Fig. 3.—Self-dumping skip, capacity 112 cubic feet.

Two pockets, each with a capacity of 300 to 500 tons, are provided at the shaft on each of these levels. These pockets are ordinarily used for handling the iron and silica ores. The relatively small amount of converter ore mined below the 1000-ft. level is stored in raises until it is convenient to empty one of the skip pockets for this service. In this

way no difficulty is encountered in handling three classes of ore through the two skip pockets on each haulage level.

The arrangement of the loading pockets follows the general practice in the camps of southern Arizona. Fig. 4 shows a section and a front view of one of these pockets. The ore passes from the loading pocket through an air-operated, undercut, arc gate to a cartridge of 112-cu. ft.

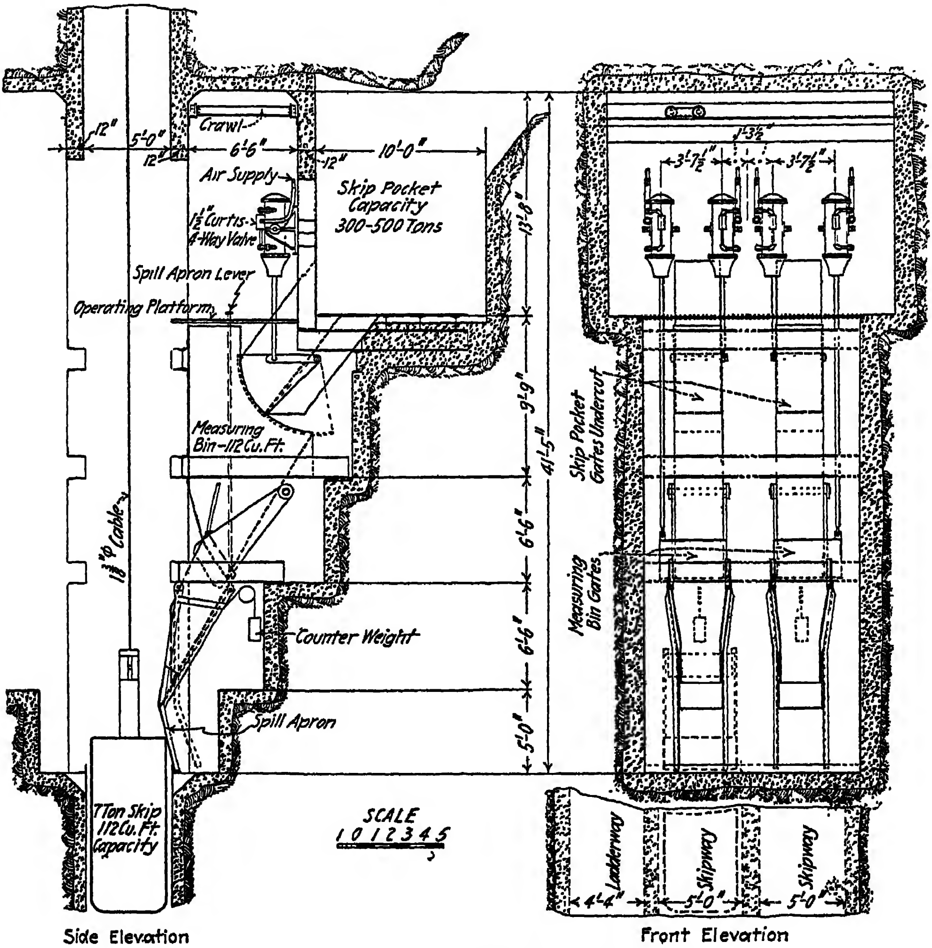


FIG. 4.—LOADING POCKET, No. 5 SHAFT.

capacity, which is a desired skipload. An interlocking valve prevents the opening of the lower, or cartridge, gate until the upper, or pocket, gate is completely closed.

From the No. 5 hoist room on the 1000-ft. level, the cableway extends at an angle of 60° to the headframe at the 800-ft. level. The hoisting ropes are supported on idlers spaced 34 ft. (10.3 m.) center to center. The sheaves are 10 ft. in diameter and operate in bronze journals.

The skips dump into a bin designed to absorb the shock of the falling ore, which then passes through an electrically driven distributer located on the 860-ft. level to any one of three chutes leading to the 1000-ft. loading bins. This distributer is controlled by the hoist operator from his platform and a series of colored pilot lights indicates the position of the distributer. At the bottom of one of these chutes there is a bypass arrangement by means of which waste hoisted through No. 5 shaft is diverted to the 1200-ft. level for use as stope filling.

No. 6 Service Shaft

No. 6 shaft, Fig. 12, is 13 by 13 ft. (3.9 by 3.9 m.) in the clear and has three compartments. The cage compartment is 8 by 13 ft.; the pipe and manway compartment is 4 ft. by 9 ft. 4 in. and the counterbalance compartment 4 ft. by 3 ft. The shaft extends from the 400-ft. level to the 1950-ft. level and is to be continued to the 2400-ft. level. It is of reinforced-concrete construction with walls of 8 in. minimum thickness and main curtain wall 12 in. thick. A concrete sublevel at the collar on the 500-ft. level permits the loading of both decks of the cage at once.

No. 6 hoist room is located on the 500-ft. level. It is 44 by 45 ft. in section and 26 ft. in height (13.4 by 13.7 by 7.9 m.). Opening off from one side is a motor-generator room 20 by 32 ft. by 13 ft. in height (6.1 by 9.8 by 4.0 m.) both rooms being lined with concrete. The cableway extends from the hoist room to the headframe at the 400-ft. level at an angle of 32°.

Nos. 3 and 4 Shafts

No. 3 shaft is being stripped of timber below the 1200-ft. level, to be used as a main ventilation raise. No. 4 shaft hoist is also dismantled and the shaft is used for ventilation purposes.

Tunnels

No. 6 shaft is connected with the 500-ft. level surface plant by the 500-ft. level tunnel. It is 1600 ft. (488 m.) long, 400 ft. being gunited to a depth of ½ in. to prevent slacking, and the other 1200 ft. timbered with 10 by 10 in. Oregon pine. The sets are 8 by 9 ft. in the clear and spaced on 5.5-ft. centers. A standard-gage, 60-lb. track, with intermediate rail for 18-in. gage, permits the delivery of steel and timber on special 18-in. trucks to No. 6 shaft without rehandling at the collar or level stations below. The standard gage, with trolley-type locomotives, delivered material for concrete and hoist-equipment direct to No. 6 shaft. The shift is handled to and from the collar of No. 6 shaft in standard-gage enclosed cars, each car seating 26 men.

Hopewell tunnel is 7000 ft. (2134 m.) long and is on the 1000-ft. level. It connects the ore bins of No. 5 shaft with the crushing plant and

storage bins on the Verde Tunnel & Smelter R.R., whence the ore is delivered to the smelter at Clarkdale. The tunnel is 10 by 13 ft. in section in the untimbered portions; 2700 ft. is timbered with 10 by 10 in. Oregon pine at 5.5-ft. centers. The sets are 9 ft. high with a 10-ft. cap and posts battered to give a width of 12 ft. at the sill.

Drilling and Blasting

All drilling operations are under the supervision of one engineer. His duties may be briefly enumerated as follows:

1. To keep a detailed record of all drills underground and see that they are kept in good mechanical condition.
2. To test new types of drills as they are received from the manufacturers.
3. To record and analyze the work accomplished by the drill bits.
4. To see that the type of round best adapted to the class of ground being drilled is used in all development work.
5. To carry on experiments relative to air pressures, hose, hose connections, etc.

Drills Used

The 248 Ingersoll-Leyner type hammer drill is used for practically all stoping, drifting, and crosscutting, as most of the ground on the lower levels is either tough porphyry or massive sulfide. In ordinary ground, one machine mounted on a 3½-in. (9 cm.) vertical bar is sufficient for each drift, the shoveler assisting the machine man in the set-up. In ground that requires two or more shifts to drill a round, two machines are mounted on the same bar and a chuck tender is provided to carry steel and powder for the two machines. A lighter type of mounted water drill is sometimes used in black schist and altered quartz-porphyry areas. The hand-rotated Ingersoll-Rand CC3-11 stoper is used in driving raises and the Sullivan DP-33 jackhammers are used for bulldozing boulders. Both the Sullivan DP-33 and Ingersoll DCRW 430 are used in sinking.

Types of Rounds

The pyramid cut is generally used in drift rounds in sulfide or tough quartz porphyry and the bottom cut in the schist and average ground. These drift rounds are shown in Fig. 5. In raising, a 12-hole or a 12-hole draw cut is generally used; and in massive sulfides, 18 holes are usually required. This type of round is shown by Fig. 6. In horizontal cut-and-fill stopes, holes are spaced largely according to the ground. Two rows of 8-ft. holes are drilled in a fan-shaped round from one set-up,

it being the policy to break the ground with the minimum amount of powder and do the bulldozing on the muck pile or grizzlies. Formerly, the holes were spaced from 2.5 to 3 ft. and the maximum length of hole was 6 ft. By using 8-ft. holes and spacing only to break the ground, the tonnage per drill shift has been increased from 38 to 66 tons.

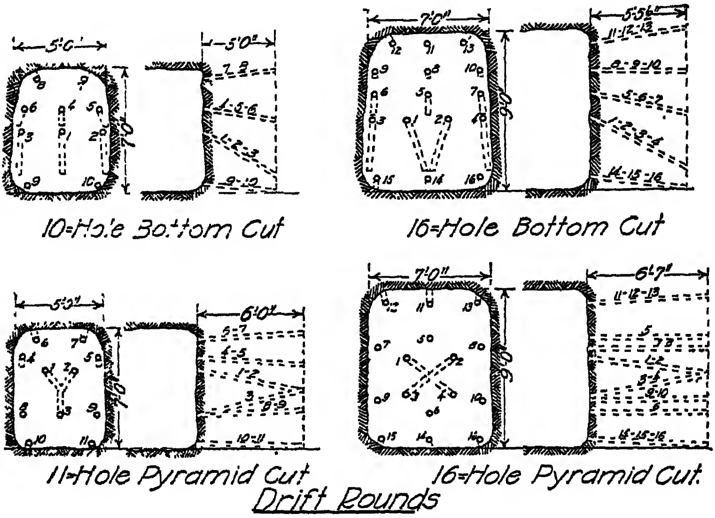


FIG. 5.—DRIFT ROUNDS.

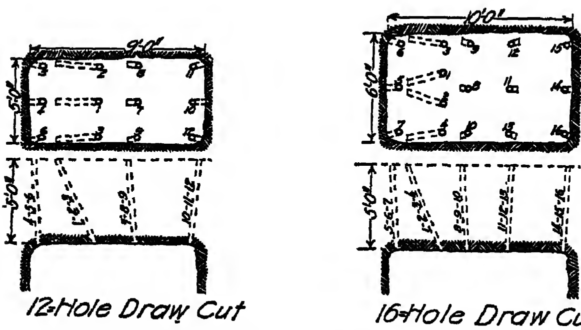


FIG. 6.—RAISE ROUNDS.

Explosives

Only two types of explosives are used in underground operations: 50 per cent. gelatin dynamite is used in massive sulfides and tough porphyry and the 35 per cent. gelatin in moderate ground. Until 1920, 35 to 50 per cent. ammonia dynamite was also used, but the slight saving in cost was offset by the ease in tamping the gelatin, a great difference in the length of "bootlegs" left, and the further advantage

of handling only two grades of explosives. These two explosives are used in the following proportions:

35 per cent. gelatin dynamite	37.5 per cent.
50 per cent. gelatin dynamite	62.5 per cent.

The powder is stored in a concrete magazine approximately 3500 ft. from the portal of the Hopewell 1000-ft. tunnel. It has a capacity of 2500 boxes, about a two months' supply for the mine. From here, the powder is hauled to the mine and distributed to the small powder magazines on each level. A powder man on each level has charge of the magazine and keeps a record of all powder given out. He also cleans the stations, makes small repairs to machines, and mends hose.

A smooth, black-finished, cotton-countered, safety fuse is now used instead of the triple-taped, gutta-percha fuse, because of the tendency of the latter to crack in cold weather. No 8X caps are used in drifting, stoping, and raising, and No. 6X are used for plugging. All fuse is cut in a special room on the 500-ft. level near the portal of the 500-ft. level tunnel. A mechanical cutter, operated by a foot pedal, cuts twenty lengths at one time. A cap crimper, operated by a foot pedal, saves time in crimping the caps. With this equipment, 800 fuses have been cut and crimped by one man in 1 hr. The capped fuses are stored in waterproof cans for distribution each day to the working levels.

Each miner does his own blasting. The primer is usually made by inserting the cap in the last stick, placing the latter so that the cap points toward the bottom of the hole. Tamping sticks, 1 in. in diameter and 10 ft. long, are used but no stemming material is generally used. Before the curtailment in 1921, a blasting crew of two experienced men did all the blasting on graveyard shift, except blockholing. Blasters are required to use tamping at all times. This system reduces the possibility of accidents in handling powder and places all responsibility on two experienced men. This crew will be reorganized in the near future.

All ore from the stopes must pass a grizzly opening 11 by 44 in. These stope grizzlies are made up of 60-lb. rails inverted and held in place on top of the stope chutes by 4 by 6 in. blocks, cut to conform with the shape of the rail. The larger boulders are drilled on the muck pile and blasted at noon, and when going off shift, the smaller ones are broken on the grizzlies. The labor charge on blockholing and plugging is 30 per cent. of the total labor charge for breaking down the ore, and the powder used is 27 per cent. of the total used in the stopes. A small amount of breaking is also done at the shaft-pocket grizzlies, where the ore has to pass a 12 by 18-in. opening before going to the skip pocket.

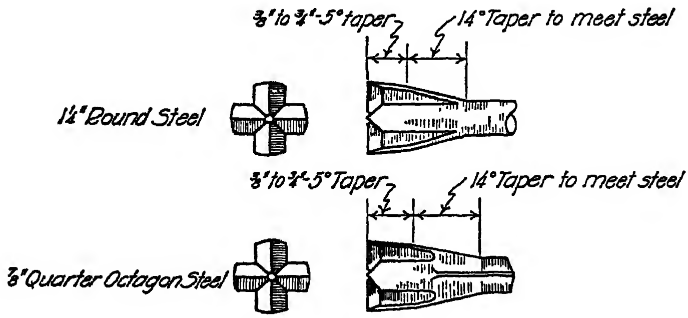
Drill Steel

Two kinds of steel are used. These are the 1¼-in. (3.2 cm.) hollow-round steel for the heavy Leyner type of water drills, and ⅞-in. (2.2 cm.)

hollow quarter-octagon steel for light jackhammers and the wet stoper drills. The size of bits is the same for both kinds of steel. Fig. 7 shows the length of standard changes.

Types of Bits

Experimental work by H. W. Seamon, who has charge of all drilling operations, has proved the suitability of the double-taper bit with $\frac{1}{16}$-in. gage changes for the hard ground encountered in the United Verde mine.



Standard Changes

Steel	Short Starters	Long Starters	Seconds	Thirds	Fourths	Finishers	Long Finishers	Extra Long Finishers
Length 1¼" Rd	12"-18"	18"-27"	27"-36"	36"-45"	45"-54"	54"-63"	63"-72"	Over 72"
Guage	1 15/16"	1 7/8"	1 13/16"	1 3/4"	1 11/16"	1 5/8"	1 9/16"	1 1/2"
Length ⅞" Q.Oct	12"-18"	18"-30"	30"-42"	42"-54"	54"-66"	66"-78"	78"-90"	Over 90"

FIG. 7.—DOUBLE-TAPER CROSS-BIT.

Table 5 shows the increase in drilling speed made by the adoption of the double-taper bit with $\frac{1}{16}$-in. changes over the single-taper bit with $\frac{1}{8}$-in. changes that was formerly used.

TABLE 5.—*Effect of Bit Design on Drilling Speed*

	Average Drilling Speed, Inches per Minute			Per Cent. Increase in Drilling Speed 3 Over 1
	1 Single-taper ⅛-in. Change	2 Double-taper ⅛-in. Change	3 Double-taper $\frac{1}{16}$-in. Change	
Massive sulfides	1.53	3.30	4.20	174
Jasper	2.65	3.75	5.20	96
Porphyry	5.10	5.48	7.88	54
Gage of starter	2¼	2¼	1 15/16	

HORIZONTAL CUT-AND-FILL METHOD

As shown in Table 4, approximately 60 per cent. of the tonnage is derived from cut-and-fill stopes. Fig. 9 shows the application of this

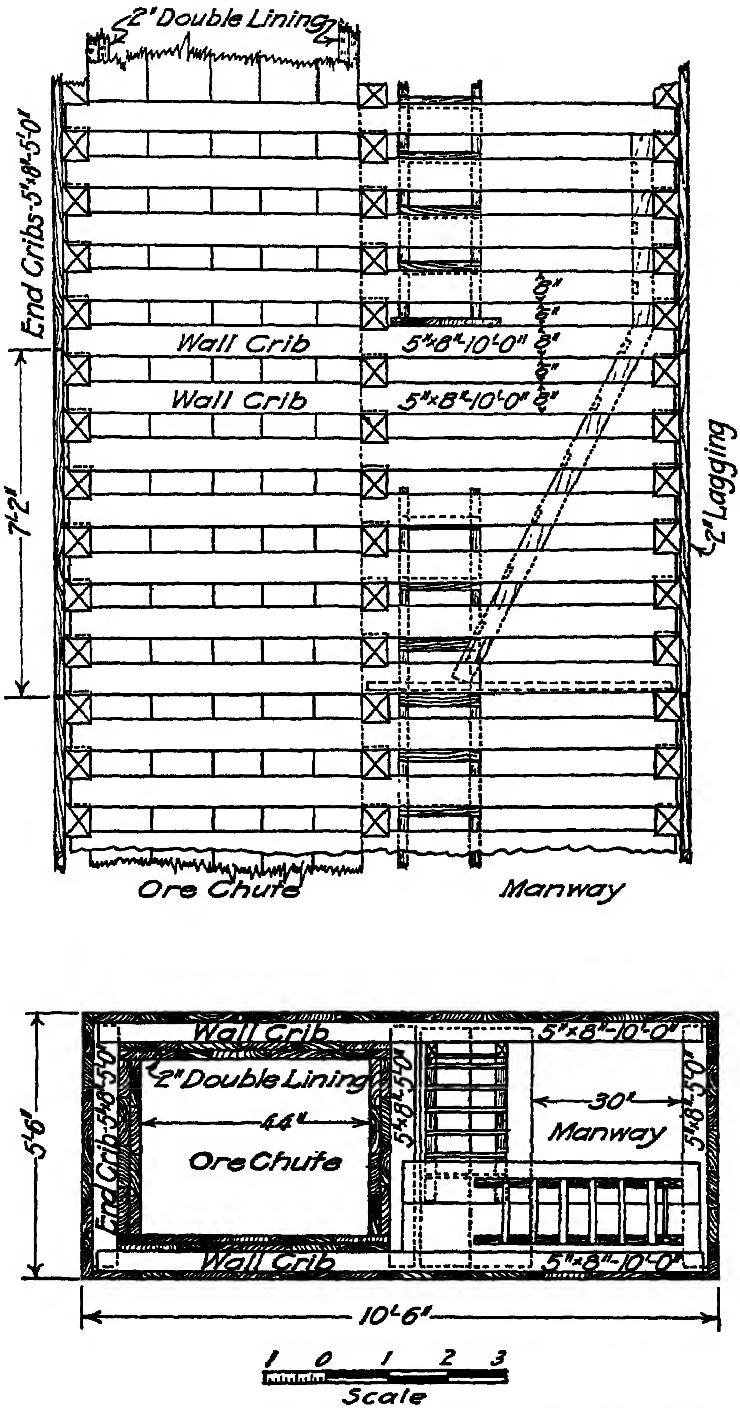


FIG. 8.—STANDARD CRIBBED CHUTE AND MANWAY.

method to an orebody 70 ft. in width by 450 ft. in length, which is shown in plan on Fig. 1.

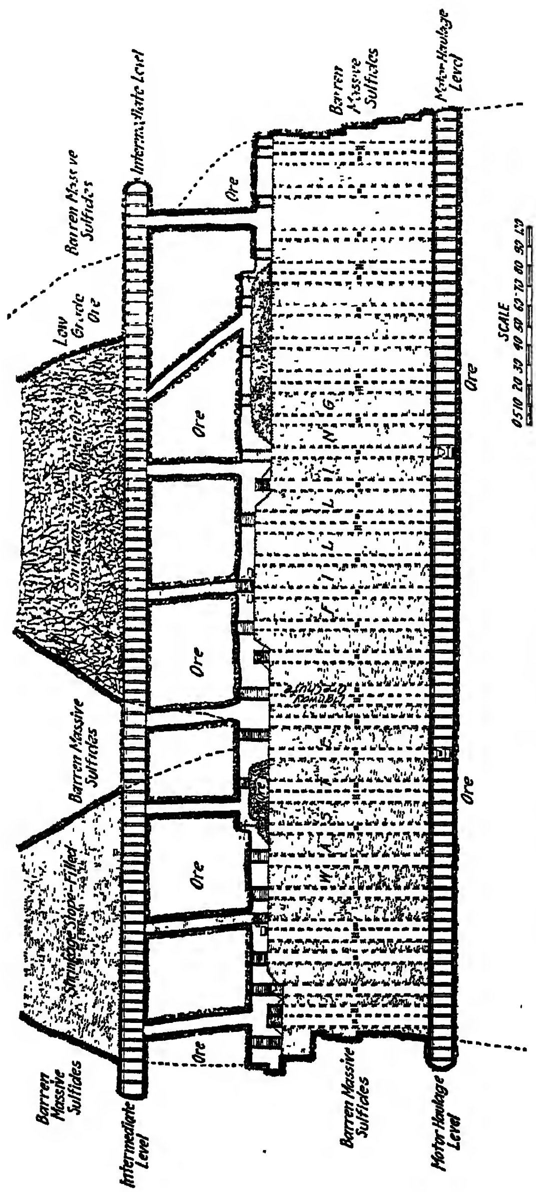


FIG. 9.—LONGITUDINAL SECTION, HORIZONTAL CUT-AND-FILL STOPE.

The first step in the development of an orebody, after its position has been determined by a crosscut, is to drive drifts to ascertain its longitudinal extent. From these drifts, by means of short raises, cross-cuts are driven on the second floor, 13 ft. (3.9 cm.) above the level, at

intervals of 100 ft. (30 m.) to determine the outline of the orebody. This work, if done on the level, would later interfere with the desired locations of gangways and make the ground heavy about them.

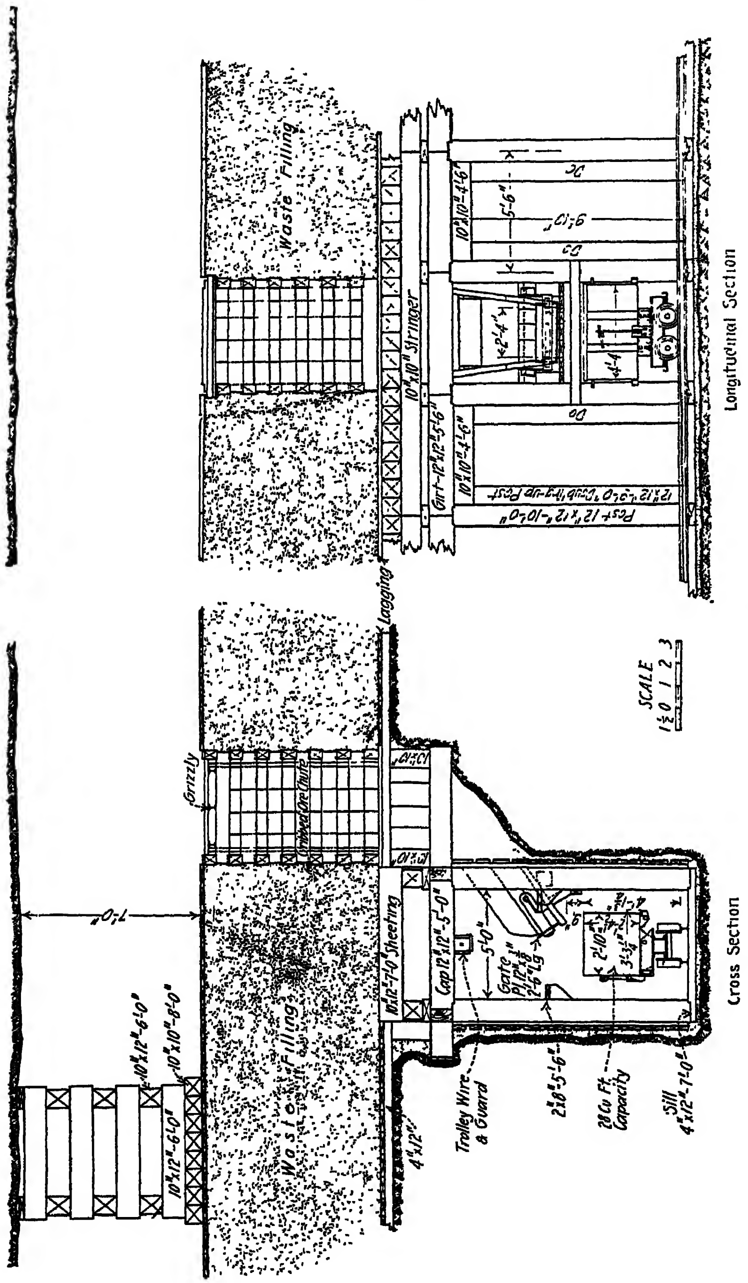


Fig. 10.—Chute and Gangway Timbering.

The next step is to enlarge the drift to gangway size, about 8 by 15

gangway timbers and the method of bridging over the gangways, Fig. 10, are designed to throw the entire weight upon the girts in the set. The 12 by 12-in. caps act chiefly as spreaders. Doubling-up posts and girts, 10 by 10 in. in section, are placed under the girts. If necessary, a solid row of posts can be used to support the girt and bridging. This method does away with the necessity of using doubling-up posts under the caps, an undesirable method in that it seriously reduces the gangway clearances.

While the gangways are being timbered, ventilation raises, approximately 200 ft. apart, are driven to the level above. These raises are located so that they may be used as waste raises in later stoping operations. Additional gangways are driven 30 ft. apart and parallel to the first gangway. Chute pockets are located in every second set on alternate sides of the gangway, which gives a spacing of 11 ft. between them. Manways are located 100 ft. apart in each gangway. They are offset in the solid from the gangway to avoid the possibility of a man stepping in front of an approaching train as he descends from a stope, and also to provide storage for steel and timber when hoisting to the stope. Solid ground is left at the bottom of all chutes, and a chute mouth is installed, as shown in Fig. 10.

Stope Silling

When the gangways have been timbered and sheeted, the stope is started on the second floor. The sill floor is started 13 ft. above the level, leaving the ground on the level intact, except for the development work described. Formerly the stopes were silled on the levels, but it was found that the side thrust on the gangway timber, transmitted through the waste filling, resulted in excessive maintenance costs.

A cut 7 to 9 ft. (2 to 2.7 m.) high is taken, using mounted hammer drills and horizontal holes. No air-feed stopers are used in these stopes as the vertical or steeply inclined holes produce a dangerous shattering of the back. As the silling proceeds, bulkheads of Oregon pine, 10 in. by 12 in. by 6 ft. (0.25 by 0.3 by 1.8 m.) are erected and spaced according to the strength of the back. Two-member bulkheads are used for average ground and three-member bulkheads, spaced as closely as efficient mucking conditions will permit, for heavy ground. In heavy ground, it is sometimes necessary to bridge between bulkheads with 10 by 10-in. timbers. On completion of the sill, or second floor, 4 by 12-in. bottom sills are laid and covered with 2-in. flooring, in preparation for a second cut of 7 feet.

Waste Raises

Simultaneously with the silling, additional waste raises are driven. These raises are 6 by 11 ft. in section and are spaced so that each raise will serve approximately 3000 sq. ft. of stoping area. The waste raises

are close to the stope walls, as raises driven in the center of the orebody weaken the back of the stope and add unnecessary danger to stoping operations.

Routine of Stoping Operations

After completion of the preliminary work, a second cut of 7 ft. is started from one of the waste raises. The weakest, or heaviest, point in the stope is selected for starting this cut, and the strongest ground left, whenever possible, until the finish of the cut. Large hammer drills of the 248 Leyner type are used for all drilling in the massive sulfide stopes. In schist stopes, the lighter mounted water drills can be used to advantage. It is the policy to break the ore with as few holes as possible and do the blockholing on the muck pile. A fan-shaped round is drilled from a vertical column and the spacing of the holes is left largely to the judgment of the miner.

Although the average drilling speed of the massive sulfides is 3.7 ft. per min., compared with 5.5 ft. per min. for the schist, the average footage drilled per shift and the average tonnage per drill shift are less in the schist stopes than in the massive sulfide stopes. This is largely because the schist stopes are generally worked by the square-set method.

Class of Ore	Average Footage Drill Shift	Average Tonnage Broken per Drill Shift
Massive sulfides	45	57
Schist	30	30

Formerly two roof inspectors, who were skilled miners, inspected the back of each stope and barred down any loose pieces of rock left by the miners. At present, the miners do their own barring-down and are responsible for the safe condition of their working place. No shoveling is permitted under the brow being drilled and a 20-ft. space in advance of the brow is kept roped off from the rest of the stope.

The shovelers work on the 2-in. (5 cm.) flooring, which serves the double purpose of keeping the broken ore and the waste fill separate and providing an easy floor for shoveling. A No. 2 square-point shovel, made from chrome-nickel alloy steel, is the standard for all shoveling. The capacity of this shovel is 300 cu. in. (4916 cu. cm.) or 23 lb. (10.4 kg.) of average sulfide ore. A No. 2 round-point shovel is used in all drifts and crosscuts where the shoveling is from a rough bottom. The capacity of this shovel is 250 cu. in. (4096 cu. cm.) or 20.6 lb. (9.3 kg.) of average sulfide ore. The two methods of transfer to the ore chutes are direct shoveling into the chutes whenever possible and by wheelbarrows for distances over 15 ft. (4.5 m.). Recently two-wheeled cement buggies, reenforced with ¼-in. plate, were tried in place of wheelbarrows and have given satisfaction. The main disadvantage of this buggy is the necessity of a smooth floor or a double track of plank. The output per shoveler

varies from 10 to 16 tons per shift, which includes the delays for breaking boulders to pass the grizzly openings. Barren country or like material, when encountered in shoveling, is sorted and thrown back, to be covered by the advancing filling.

The ore chutes, which are carried up through the stope as mining proceeds, are built of native pine cribbing 5 in. by 8 in. by 5 ft. and lined with two thicknesses of 2-in. lagging on the inside and one thickness of used flooring on the outside. The single chute is shown by Fig. 10 and the double chute and manway by Fig. 8. Recently 6-in. lining, beveled on each end and held in place by key blocks, has been tried and proved successful. This type of lining has the advantage of being more easily placed and outlasts three 2-in. thicknesses. It is easily placed in new chutes but unsatisfactory for relining old chutes. Chutes are spaced approximately 25-ft. centers, and if the gangways are more than 30-ft. apart, the chutes are offset in the waste fill by means of square sets.

Short bulkheads are built upon the broken ore, and bulkheads 14 ft. (4.2 m.) in height are erected from the shoveling floor to the back of the stope. As soon as the shovelers have provided sufficient cleared space, waste filling is dumped from the level above into the waste raise until no more filling will run into the stope. The cone formed at the bottom of the raise is then leveled off to a height of 7 ft. above the floor and a temporary chute is constructed. The filling is then distributed by tramming in cars of 18-cu. ft. (0.5-cu. m.) capacity, until the next raise becomes available. Sectional stope track in 8-ft. lengths and curves with 9-ft. radius are used in all stopes. These are made up of 16-lb. rail riveted to 3/8 by 4-in. steel-plate ties and are connected by means of a slip-joint tie and held together by a rail spike, no bolts being necessary.

As the shovelers advance, the short bulkheads on the broken ore are taken down and 14-ft. bulkheads are built up from the floor of the stope. Similarly, as the filling advances the 14-ft. bulkheads are removed and replaced by short bulkheads built upon the waste. When the ground is heavy, the long bulkheads are often left standing and the filling placed around them. All bulkheads on filling are erected upon a mat of double 2-in. (5 cm.) lagging, or 10 in. by 10 in. by 8-ft. timber, in order to transmit the pressure to a large area of the filling. As soon as the cribbed ore chutes are not needed by the shovelers, they are built up to the next floor and filling is placed around them.

The filling is covered by a flooring of 2-in. lagging and is ready for the next cut. This lagging is taken up as the filling advances; it is used again as flooring and then used for lagging the outside of chutes or as pillar lagging to separate the filling from the ore pillars (40 ft. wide) left between large stopes.

In all stopes having an area of 10,000 sq. ft. (920 sq. m.) or more, it has been found economical to install an air hoist on the level with a

light single-deck cage for handling men and supplies to the stope. In smaller stopes, an Anaconda stope hoist is generally used. The mounted drill-column hoists are used almost exclusively for raise work because of their slower rope speed, lighter weight, and portability.

When a horizontal cut-and-fill stope has been carried up to within 20 or 30 ft. of a level, the remaining floor pillar is mined in small sections by overhand square-set methods.

Incline Cut-and-fill Method

The incline cut-and-fill method is seldom used in the United Verde mine. It is applicable to narrow stopes free from water courses where the walls are not strong enough for shrinkage. In one of these orebodies, 35 ft. wide by 180 ft. long (10.6 by 54.8 m.) the operation was as follows:

A drift was driven along the main axis of the orebody and the usual gangway timbering and chutes were erected. At the center of the orebody, a waste and ventilation raise was driven to the level above. Inclined overhand cuts 7 ft. wide were then taken alternately on each side of the raise. The central location of the raise permitted the steady operation of the stope, one being filled with waste while breaking ore was in progress in the other.

This method was successfully used in removing the floor pillar in a large cut-and-fill stope where the back was strong and the stope on the level above had been mined by shrinkage but not filled. The stope was divided into two sections and the successive inclined cuts were taken, beginning at the extremities of the stope and working back toward the center where the main crosscut on the sill above was located. The miner was protected from small rocks that might fall from the back of the open stope above, by working under the brow of the inclined cut, and the waste trammer, by carrying drift sets along with the advancing fill and carrying the fill ahead in the shrinkage stope above.

Square-set and Fill Stoping

The overhand square-set stoping method in use at the United Verde mine differs but little from standard practice for this method. Gangways are timbered in the same manner as for a horizontal cut-and-fill stope. All sets are 5 ft. 6 in. (1.67 m.) square and 7 ft. 2 in. (2.2 m.) high. The regular square-set chute and manway have the same dimensions as the cribbed chute and manway shown in Fig. 8.

The square-set method is used only in mining heavy ground, in coming up under completed stopes, and for stope pillars, so that it is always necessary to keep the minimum amount of ground open. Consequently, only a few sets more than the mining floor are left unfilled at any time. This condition prevents the advantageous use of slides and necessitates

the shoveling of practically all the broken ore into chutes spaced 22 ft. from center to center.

Shrinkage-and-fill Stoping

This method is applicable to mining of orebodies of massive sulfides when the walls consist chiefly of low-grade sulfides. The development and sill-floor work for shrinkage stoping is similar to that described in detail for the horizontal cut-and-fill method. After the sill floor has been mined, ore breaking proceeds in 7-ft. cuts, as in horizontal cut-and-fill work. As many boulders as possible are blockholed after each round to avoid the costly blasting in the chutes upon the levels. Cribbed chutes and manways spaced 100 ft. apart are used for ventilation and supplies. Occasionally 10 by 12-in. bulkheads are built upon the broken ore where bad ground is encountered. No other timer, except as noted above, is used in this method. On account of its many advantages and low cost, this method is used wherever practicable.

Shrinkage Stoping with Pillar Caving

To meet the smelter requirements for free silica, an orebody in quartz-porphyry schist carrying 1.5 per cent. copper as secondary chalcocite was prepared for mining. This orebody contains approximately 1,000,000 tons and extends from the 160-ft. to the 600-ft. level. It is roughly pear shape in outline and is about 250 ft. (76 m.) long and 150 ft. (45 m.) wide. The gradation to barren porphyry is gradual on all sides of the orebody.

A combination of the Ray system of narrow shrinkage stopes and pillar caving with the Inspiration method of drawing through branched raises was selected. The motor-haulage gangways were located on the 600-ft. level and the grizzly drifts 40 ft. above. After mining approximately 75,000 tons of ore by this method, operations were suspended, silica now being obtainable by steam-shovel operations.

Glory-hole Stoping

Previous to steam-shovel operations, the oxide area above the 160-ft. level was mined by glory-hole methods. Raises were driven from this level to the surface at intervals of approximately 60 ft. and served as starting points for underhand stoping with unmounted hammer drills. The ore was lowered through No. 4 shaft and dumped into the bins leading to the 1000-ft. haulage tunnel, the oxide being too sticky to handle through raises.

A system of raises has been completed from the main ore raises leading to the 1000-ft. bins to the surface. These raises serve for steam-shovel ore, which is handled through the mine, and certain ones on the extremities of the pit limits will be used for glory-hole or mill-hole mining. Mill-hole

mining will be carried along in connection with the steam-shovel operations down to the 200-ft. level and the comparative costs will largely determine the method used in mining as far down as the 400-ft. level.

Mining of Waste for Stope Filling

The waste fill for stopes is secured from a surface glory hole with a main waste raise 9 by 20 ft. (2.7 by 6 m.) in section, leading to the 1800-ft. level. A grizzly and blasting chamber is located above the 500-ft. level, the grizzly being made up of double 10-in. I beams with wearing plates and set with 3-ft. openings. A second blasting chamber is at the 1000-ft. level with a grizzly made up of Morrison guard rails with 13-in. openings. Drawing-off chutes and dumping raises are provided at each level so that waste may be drawn off or dumped from any level, as required. The waste is handled from the drawing-off chutes to the stope waste raises by storage battery or trolley motors.

Operations

Timbering

In the United Verde mine, Oregon pine is used for all permanent drift timbering and bulkheads, whereas native pine is used for lagging, cribbing, flooring and square-set stope timbering.

Fig. 10 shows the details of a standard gangway set with dimensions. These timbers are so framed that the weight of the filling is taken by the girts, the caps acting merely as spreaders. The drift sets are similar to the gangway set, except that 10 by 10-in. timber is used and the posts are 7 ft. 7 in. over all.

Fig. 8 shows a double cribbed chute which is carried up through the stope fill. The single chutes, without manways, are built in the same manner, except that single cribbing, 5 ft. long, is used. Cribbing 4 in. by 6 in. by 4 ft. is used as manways in raise work and later stripped and used again or cut up for mine ties.

Bulkheads are of the two- or three-member type, as shown in Fig. 10. They are made up of Oregon pine 10 in. by 10 in. by 8 ft. laid directly upon the broken ore, or upon a double layer of 2-in. flooring where built upon fill.

Chute lining and flooring comes in all widths and 2-in. thickness. The chute lining is 7 ft. 2 in. long and the flooring 5 ft. 4 in. in length. Flooring and bulkhead material are used until they have no further value underground. They are then sent to the surface and used in the heating plant.

All timber is framed at the mill and unloaded directly to the proper place in the storage or lumber yard, which is 1000 ft. from the portal of the 500-ft. level tunnel. Each day, the foreman gives a list of the timber

requirements for the following day to the lumber-yard foreman and the timber is loaded on special 18-in. gage timber trucks and delivered to No. 6 shaft by storage-battery locomotives on the afternoon shift. These timber trucks are delivered to the working place without any transfer.

Preservatives are not used to any extent, although it is proposed to use gunite on timbered drifts in the converter orebody, which will not be mined for several years.

Underground Sampling

The sampling department consists of two men under the supervision of the geology department. All stope and development in mineralized areas are sampled as the work progresses. The usual method of sampling is to take a 5-lb. sample by means of a prospector's pick; in special cases a groove or channel sample is taken. In stopes, each sample represents a block 5.5 ft. by 5.5 ft. by 7 ft. 2 in., or what is termed a "set." This set is the unit of extraction and all engineering records on stopes give the tonnage in sets rather than tons. In drifts and crosscuts, three samples are taken—left, face, and right—at 5-ft. intervals, except where the formation is known to carry no values.

Loading Machines

Scrapers, Armstrong Shuveloders, and an electrically driven Marion shovel, type No. 28, were tried with little success at this mine. The massive sulfide breaks in large pieces and is so heavy that mechanical muckers have shown no saving over hand work.

Shuveloders have proved successful in development work, where the ground breaks much finer than in stoping, and will be used in main haulageways where it is not necessary to increase the size of the drift merely to accommodate the machine.

Tramming and Haulage

Trolley type of motor haulage is used on all main haulage levels, and storage-battery locomotives on the intermediate levels. Hand tramming is used only where the tonnage does not warrant the changes necessary to use storage-battery locomotives. Formerly, storage-battery locomotives hauled the ore on the intermediate levels, an average distance of 300 ft. to transfer raises. From here, it was taken to the ore pockets on the level below by trolley motors. It was found more economical to extend the ore pockets from the main levels to the intermediate levels and eliminate the transfer, even though the length of haul for the storage-battery locomotive was increased from 300 to 1000 ft. The motors can be taken from one level to another on No. 6 shaft cage. Storage-battery

locomotives also handle all timber, steel and supplies from the 500-ft. level surface plant to the collar of No. 6 shaft.

The 250-volt direct current for motor haulage is carried by No. 0 wire, except in Hopewell tunnel, where No. 000 is used. Wherever the wire is less than 8 ft. (2.4 m.) above the rail, it is guarded on each side by 1¼ by 8 in. boards. A hinged board, 5 ft. long and fastened to one guard, is provided at all chutes. When hooked to the opposite side by the trammer, when loading cars, this board completely covers the live trolley wire, preventing accidents.

Six-ton, Jeffrey, trolley-type, mine locomotives, with solid-slab frames, are standard for the main haulage levels. These locomotives are provided with two 18-hp. motors, which are connected to the drive wheels by intermediate gearing because of the narrow gage of the track. Steel-tired wheels and ball-bearing armatures aid in maintaining a low upkeep cost. The rated drawbar pull is 3000 lb. at a speed of 3 mi. per hr. on a level track. A train of twenty 18-cu. ft. cars is used for ordinary haulage, with a trailing load of 35 tons when hauling massive sulfide.

A 3-ton Baldwin-Westinghouse, storage-battery locomotive equipped with 80 Edison type A-4 cells, is used on the intermediate levels. This motor has a drawbar pull of 800 lb. at 3½ mi. per hr., a speed sufficient for short hauls. The batteries are changed and given a boosting charge each shift.

Baldwin-Westinghouse, 25-ton, trolley-type locomotives are used in Hopewell tunnel. These locomotives are equipped with two 75-hp., commuting-pole type motors and automatic air brakes. The drawbar pull is 12,500 lb. at 7.1 mi. per hr. All ore hoisted by No. 5 shaft is dumped into storage bins above the 1000-ft. level and hauled in standard-gage bottom-dump cars of 220 cu. ft. nominal and 280 cu. ft. loaded capacity, a distance of 8900 ft. to the Hopewell crushing plant. The average gross weight of the 10-car loaded train handled by each locomotive is 279 tons. The cost per ton-mile, including loading at the bins, supplies, power, and repairs, is $0.079. These cars will be replaced by side-dump cars of 40-ton capacity, to handle increased tonnage resulting from steam-shovel ore passed through the mine.

Cars and Track

The size of car used for all hand-tramming and motor-haulage levels was limited until recently, by the size of the cages used in No. 3 shaft. This car is a typical side-dump car with a capacity of 18 cu. ft. (0.5 cu. m.). It can be dumped without uncoupling from the train and without stopping the train. With the introduction of skip hoisting, a car of somewhat similar type has been designed with a capacity of 28 cu. ft. and a dumping arrangement similar to the Granby car. The dimensions of these two cars compare as follows:

	Inside Dimensions, in Inches Length	Width	Depth	Over All, in Inches Length	Width	Depth	Total Weight, Pounds
18-cu. ft. car....................	44	30	24	49	35	45	960
28-cu. ft. car......................	52	34	28	59	39	50	1600

A 12-in. car wheel with a 3¼-in. tread of chilled cast iron and solid roller bearings of the Sanford-Day type has been adopted as standard.

The use of 16-lb. (7.25 kg.) rails, ties 4 in. by 6 in. by 3 ft. (10 cm. by 15 cm. by 0.9 m.) and 18-in. gage is standard throughout the mine for all hand tramming. Motor-haulage levels are equipped with 40-lb. rails, ties 6 in. by 6 in. by 3 ft. and 18-in. gage. Stub switches are used for both sizes of rails and manganese-steel frogs on main-line haulage lines. In haulage levels, curves are planned with a minimum radius of 50 ft. The grade is 0.4 per cent. in favor of the load.

Underground Storage and Dumping

Loading pockets are provided at No. 5 shaft on the main haulage levels, at intervals of 300 ft. These pockets, of 500-ton capacity, follow the general practice in the camps of southern Arizona. Fig. 4 shows a section and front view of one pocket. The ore passes from the loading pocket, through an air-operated, undercut, arc gate to a cartridge of 112 cu. ft. (3.1 cu. m.) capacity, which is the desired skipload. An interlocking valve prevents the opening of the lower, or cartridge, gate until the upper, or pocket, gate is completely closed.

The skips (Fig. 3) of 112 cu. ft. capacity dump into a bin, so designed as to absorb the shock of the falling ore. This ore then passes through an electrically driven selector to one of three storage bins, depending on the class of ore being hoisted. Colored pilot lights indicate the position of the distributer, which is on the 860-ft. level and is controlled by the hoist operator from his platform. These main ore bins are above the 1000-ft. level, and have a capacity of 1000 tons each. From these, the ore is passed through vertical-gate chutes, operated by 12-in. air cylinder, to the Hopewell cars.

Hoisting

No. 5 shaft hoist is an Allis Chalmers, double-drum, single-reduction, geared type, driven through a flexible coupling by a 650-hp. d.c. motor, with current at 500 volts and a normal speed of 300 r.p.m. It is designed to operate two skips in balance to a depth of 2200 ft. below the 1000-ft. level. The maximum rope speed is 1000 ft. per min. and the maximum capacity is 5500 tons per 24 hr. In emergencies, the hoist will operate unbalanced for a period of 4 hr. Full speed is attained in 10 sec. and retardation to rest in 5 sec.

The drums are cylindrical, 10 ft. in diameter, with a 5-ft. smooth face (3 by 1.5 m.) and will hold 2500 ft. of 1⅜-in. (35 cm.) extra plow steel

6 by 19 hoisting rope, in two layers. Each drum has a double-disk, multiple-arm, friction clutch and a parallel-motion post brake, which are operated by auxiliary engines actuated by oil under pressure. Power is supplied to the hoist motor by a 695 r.p.m. flywheel motor-generator set, consisting of one 700-hp., wound-rotor, induction motor, one 600-kw. 500-volt, d.c. generator, one 10-kw. 250-volt exciter, and a steel-plate flywheel 9½ ft. (2.85 m.) in diameter, weighing approximately 40,000 lb. The starter and slip regulator are of the standard liquid type, designed to start the set from rest. The regulator varies the resistance in the secondary circuit of the motor as the service demands, permitting the flywheel to deliver or store energy as required.

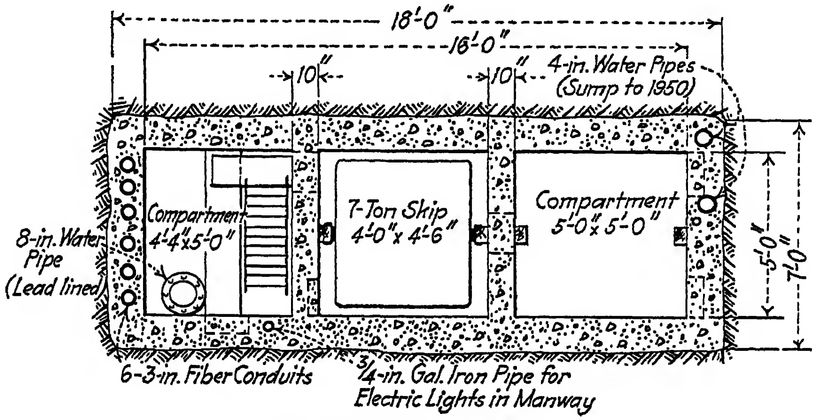


FIG. 11.—No. 5 SHAFT.

In addition to an overwinding device on the hoist indicator, two limit switches in the headframe are operated by the upcoming skip and automatically stop the hoist if the overwinding device fails to function. An emergency switch, placed near the operator, cuts off the current and applies the brakes.

The skips (Fig. 3), which operate in balance, are of 112 cu. ft. capacity (3.1 cu. m.), equivalent to 8 tons of iron ore, 65 tons of silica ore, and 5.8 tons of waste. They are of rugged construction and weigh approximately 12,000 lb. (5443 kg.).

No. 6 Hoist.—No. 6 hoist is designed to operate a double-deck cage, in balance with a counterweight, at a rope speed of 800 ft. per min. The acceleration is 6 sec. and the retardation 5 sec. The hoist is of Nordberg manufacture and has a single drum 12 ft. (3.6 m.) in diameter with a 6-ft. smooth face holding 2500 ft. of 1¾-in. (1.9 cm.) extra plow steel rope in two layers. A reel for ½-in. by 5½-in. flat rope for the counterweight is rigidly coupled to the main drum. The brakes are of the parallel motion, gravity, post type, set by a deadweight and released by a hydraulic thrust cylinder operating under oil pressure. The hoist

is driven through a flexible coupling by a 350-hp., wound-rotor, induction motor at a normal speed of 360 r.p.m.

The cage is of double-deck design, built by Wellman-Seaver-Morgan Co. It is approximately 24 ft. high, each deck is 6 ft. 8 in. by 12 ft. 7 in. and will accommodate from 60 to 70 men on each deck. Two 18-in. gage tracks give each deck a four-car capacity. The cage is fitted with double gates, which have offset hinges and prevent the gate from swinging outward. The weight of the cage (9½ tons) and the load is balanced by a counterweight weighing 15.7 tons.

Besides the usual safety dogs on the cage, the hoist is provided with an overspeed safety device, which throws off the power and applies the brakes in case the cage should attain a speed of over 800 ft. per min. The shaft is also provided with limit switches at both the upper and lower extremities. These also cut off the power and apply the brakes in case of over-winding.

Pumping

The average flow of water in the mine is approximately as follows:

	Gal. per Min.
Copper-bearing water above 1000-ft. level	17
Barren water above 1000-ft. level	36
Barren water below 1000-ft. level	52
Total	105

The amount of water handled above the 1000-ft. level is greatly in excess of these figures in periods of heavy rain or snow. The water below the 1000-ft. level that must be handled by pumps seldom exceeds 60 gal. per min. The pumping problem at the United Verde mine has, therefore, been a simple one. A 50,000-gal. sump has been provided on the 1950-ft. level by driving the shaft crosscut 5 ft. (1.5 m.) below grade for a distance of 300 ft. and then concreting this section to track grade.

The pumping equipment on the 1950-ft. level consists of an electrically driven Aldrich quintuplex pump, 6½ by 12 in., direct-connected to a 100-hp., 430-r.p.m., 2200-volt, slip-ring induction motor. This pump has a capacity of 500 gal. per min. under 1000-ft. head, and pumps directly to the 1000-ft. level through an 8-ft. lead-lined pump column.

The air pump, which is used for auxiliary purposes only, is a double-action plunger-type, 32 by 10 by 24 in. (80 by 25 by 60 cm.) stroke, with a capacity of 300 gal. (1135 l.) per min. at 90 lb. (40.8 kg.) air pressure.

Both pumps are fitted with bronze rods, bronze-lined pistons, and bronze-lined stuffingbox glands to resist the action of the mine water, which carries a small amount of free acid.

The water is handled from one level to another through diamond-drill drainage holes, 2³⁄₁₆ or 2 ¹³⁄₁₆ in. in diameter.

Air Compression

The compressor plant consists of three electrically driven Ingersoll-Rand compressors, all of which are the two-stage cross-compound type, delivering air at 100 lb. No. 1 is of the PB-2 class with cylinders 18¼ by 30¼ by 24 in. stroke (46.3 by 77.3 by 61 cm.) with a displacement of 1500 cu. ft. of free air per minute. It is belt driven by a 300-hp. General Electric induction motor operating on a 60-cycle, 2080-volt, 77-amp. alternating current.

No. 2 compressor is also of Ingersoll-Rand manufacture, with cylinders 19¼ by 31½ by 30-in. stroke (48.8 by 80 by 76.2 cm.) with a displacement of 3000 cu. ft. of free air per minute at 5000-ft. elevation. It is driven by a General Electric 600-hp. induction motor, operating on a 2200-volt, 60-cycle, 144-amp. alternating current.

No. 3 is of later design, being an Ingersoll-Rogler type PRE-2, 28 by 46 by 36-in. stroke (71.2 by 116.8 by 91.4 cm.) with a displacement of 6800 cu. ft. of free air. It is direct-connected to a synchronous motor manufactured by the General Electric Co. It is rated at 1163 hp. and operates on a 2200-volt, 60-cycle, 243-amp. alternating current. The exciter is a Westinghouse, Type CS motor, 38.5-hp., 2200-volt, 60-cycle, direct-connected to Westinghouse shunt-wound generator 25-kw., 125-volt, 200-amperes.

Ventilation

A Sirocco fan located just off of Hopewell tunnel, near No. 6 shaft, supplies air to the entire mine. This fan is a No. 15, 90-in. (229-cm.) single-inlet pressure fan with a capacity of 175,000 cu. ft. (4900 cu. m.) driven by a 400-hp., wound-rotor, induction type motor operating on 2200 volts. A series of resistances in the secondary circuit makes it possible to control the speed of the motor and regulate the amount of air delivered by the fan.

The air is drawn into Hopewell tunnel, also through the 500-ft. level tunnel by way of No. 6 shaft. The intake tunnel to the fan is 120 sq. ft. in section and the air is delivered under 4-in. (10 cm.) water gage to a ventilation drift of the same section, and then splits in drifts and raises of considerably greater total section for general distribution to the mine workings. The air for the levels below the 1000-ft. level is forced down through the stopes and outlets to the shafts on the lower levels. A more efficient method will soon be adopted by admitting air to each level or certain lower levels and allowing the air to rise through the stopes.

No. 3 shaft is being stripped of timber, giving a 7 by 17-ft. section raise to deliver air from the fan to the lower levels. An 8 by 15-ft. inclined raise has been driven from the 1200-ft. level to the discharge side of the fan and the shaft bulkheaded below the 1000-ft. sill. At each shaft station, a concrete collar will be built around the shaft, with a 4 by

6-ft. adjustable steel door. The air to the lower levels will be forced down the shaft and a certain volume taken off at each level. It will then find its way through the stopes to the shafts, thence to surface. The direction of air is controlled by wooden doors made up of 2-in. tongue-and-groove Oregon pine hinged to concrete door frames. All air doors are hand operated, except the large doors in the main Hopewell haulage tunnel.

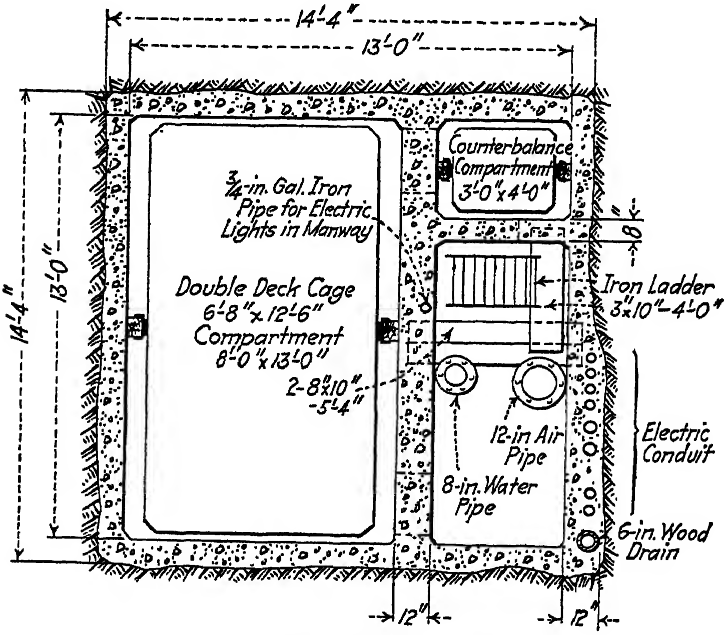


FIG. 12.—NO. 6 SHAFT.

Lighting

All shaft stations, main drifts, and crosscuts and timbered gangways are lighted by electricity. In the main drifts, 25-watt lamps are spaced at intervals of 100 ft. (30.5 m.), and in the timbered gangways where the chutes cut off the light, to a large extent, the interval is reduced to 30 ft. (9.1 m.). Lights are also placed opposite all manways and over all switches.

Miners use the ordinary Justrite carbide lamps, which fasten on the cap. The carbide is received in 100-lb. cans and is treated with coal oil to prevent slacking.

Telephone System and Signals

A magneto type mine phone, 1336-E, made by the Western Electric Co., is on each level at No. 6 shaft station and one in each hoist room. They are connected on five-party lines to the local exchange of the Moun-

tain States Telephone and Telegraph Co., as no private exchange is maintained by the company.

The pull-bell system is used in the shafts for signaling the hoist operator, and the standard code of level signals is used. For calling the cage, each station is provided with a push button, which is connected to an annunciator on the operator's platform, indicating from which level the call came. The hoist operator gives a return signal by flashing a lamp located above the push button. These return-signal lamps are connected in parallel to a single line and necessarily the return signal flashes on all levels.

Steam-shovel Mining

Most of the orebodies above the 500-ft. level are at present bulkheaded and inaccessible on account of mine fires. As the ground in this area was generally badly fractured, square-set methods were necessarily used. By the use of the plenum system of forcing air under pressure into the fire stopes, it was possible to mine this ore, but at an extremely high cost.

To make possible the mining of the ore on these upper levels, it was decided to use a combination of steam-shovel and mill-hole methods. The present plan calls for the removal of 7,790,000 yd. (5,950,000 cu. m.) of material down to the 400-ft. level. It is estimated that approximately 1,770,000 yd. of direct-smelting ore will be recovered to this depth. In addition, approximately 350,000 cu. yd. of possible milling ore will be stock-piled; also 1,120,000 yd. of lower grade non-commercial ore will be stockpiled for possible recovery in the future.

Steam-shovel ore is handled through a system of raises to the main loading bins above the 1000-ft. level, where it is handled in the same manner as mine ore. Oxide, due to its sticky characteristic, will be shipped over the V. T. & S. R. R. and the waste overburden hauled to waste dumps a short distance from the pit limits.

Disposition of Ore After Reaching Surface

As before stated, the ore from the main loading bins on the 1000-ft. level is hauled to the crushing plant at the portal of Hopewell tunnel by 25-ton Jeffrey trolley-type locomotives. The cars are of the bottom-dump hopper type and the dumping mechanism is operated by air. The capacity of this car is 280 cu. ft., or 19.5 tons of iron ore, 16.3 tons of silica and 13.0 tons of converter.

The direct smelting ore from the mine is dumped into a steel bin of 12,000-ton capacity, which has several sections for various classes of ore. From here, it passes through the crushing plant, which makes two products, *viz.*, 3½ and 7/16-in. sizes. The ore, after being crushed, is elevated to certain sections of the bins, which are reserved for storage, and is

then hauled to the smelter at Clarkdale, 6.7 miles distant, over the standard-gage Verde Tunnel Smelter Railroad.

The steam-shovel stockpile material is passed through one section of the Hopewell bin into 20-yd. air-dump cars built by the Western Wheeled Scraper Co. and is hauled an average distance of 1500 ft. to the stockpiles.

A new crushing plant, with a capacity of 5000 tons a day, is now under construction at Clarkdale and will replace the present one at Hopewell portal.

Bonus

A bonus system for the mines is applied to all raising, drifting, and stoping; also to muckers in stopes and development faces. This system has been used for 5 years and is based on sharing with the men any saving made as a result of any improvement in their work. Recently, the system has been changed so as to be applicable to all kinds of work, both underground and in the shops. This system is similar to that used in the Santa Fe shops; it is based on the efficiency of a man, and a chart is prepared showing the actual bonus in dollars and cents paid on all wage scales for each per cent. efficiency.

The base rate set for miners is a certain number of feet advance per machine shift, depending on working conditions and class of ground. Muckers in stopes are based on number of cars per shift and muckers in development faces are rated on daily advance per machine shift.

There are two prerequisites to the successful operation of a bonus system: (1) The base rate, upon which efficiencies are computed, must be correctly and uniformly established for each class of work; and (2) a rate once established must not be cut, no matter how advantageous it may prove to the miner.

The first requirement is met by having all bonuses set by one engineer, whose familiarity and experience with the different classes of ground and working conditions in the mine is such that he can quickly and fairly classify any new piece of work. The second requirement is met by the fixed rule that the bonus engineer is permitted to increase the bonus rate where the ground proves harder than anticipated, but under no conditions may he cut a rate once established. The average miner is suspicious of any contract or bonus system, and it is only by strict adherence to a fair policy that he retains confidence in the bonus system and endeavors to raise the standard of his work. Each day bonus cards are posted on bulletin boards in the change rooms. These cards show each man his daily performance with relation to the base rate, as set for that particular working place. In this way the workman's interest is sustained and at the end of the month, when the bonus is paid, he does not suspect that the bonus engineer has changed his rate and given him less than he should have received.

Records of Unit Production

Tonnage for June, July, and August, 1922

Class of Ore	Dry Tons	Percentage of Total
Iron	56,301	39.7
Silica	81,707	57.6
Converter	3,698	2.6
Precipitates	100	0.1
Total	141,806	100.0

Tons per Man per Hour and Man-hours per Ton

June, July, and August, 1922

For Miners in Stopes

Stoping Method	Tons per Man per Hour: Miners	Tons per Man per Hour: All Men	Man-hours per Ton: Miners	Man-hours per Ton: All Men
Horizontal cut and fill	9.05	0.72	0.11	1.39
Incline cut and fill	5.90	0.51	0.17	1.96
Square set and fill	1.84	0.46	0.54	2.18
Shrinkage and fill[a]	3.74	2.07	0.27	0.48
Glory hole	26.1	25.5	0.04	0.04

[a] Figures shown are based on ore broken in stope, not on the tonnage drawn off.

Tonnage Factors

	Cu. Ft. per Ton
Iron	8.3
Schist	9.9
Porphyry	11.5

For Miners on Development Work

	Tons per Man per Hour: Drifting 5 by 7 Ft.	Tons per Man per Hour: Raising 6 by 10 Ft.	Man-hours per Ton: Drifting 5 by 7 Ft.	Man-hours per Ton: Raising 6 by 10 Ft.
Massive sulfides	0.93	0.56	1.07	1.79
Schist	1.38	0.81	0.72	1.23
Quartz porphyry:				
Fresh	1.10	0.66	0.91	1.51
Altered	1.30	0.80	0.77	1.25

Totals for Organization

	All Underground Labor	All Surface Labor[b]	Total Organization[c] Includ. Mech. Dept.	Total Organization[c] Exclud. Mech. Dept.
Tons per man per hour	0.495	1.358	0.350	0.447
Man-hours per ton	2.020	0.736	2.858	2.237

[b] Excluding office but including Mechanical Department.

[c] Including office force.

CLASSIFICATION OF LABOR

NATIONALITY	PER CENT.	NATIONALITY	PER CENT
American	41.46	Scandinavian	0.39
Canadian	0.50	Russian	0.56
English	0.25	Serb	1.24
Irish	0.21	Slav	1.80
Mexican	44.72	Austrian	2 27
Spanish	4.12	Bulgarian	0 25
Italian	1.67	Miscellaneous	0.56

Labor Turnover

The labor turnover for the three months reported averaged 17 per cent each month. This is probably higher than it will average for the year due to the resumption of operations following the 1921 curtailment in production, which means that a larger proportion of those hired are new men who are more apt to leave than former employees, who are rehired.

EXPLOSIVES, POUNDS PER TON OF GROUND BROKEN

	MASSIVE SULFIDE	SCHIST	FRESH QUARTZ PORPHYRY
Drifting	8.8	8.9	13.1
Raising	8.0	5.9	9.7
Stoping	1.23	2.16	

	LB. PER TON
Horizontal cut and fill	1.40
Incline cut and fill	1.48
Square set	1.83
Shrinkage	0.70
Glory hole	0.53

Total explosives per ton of ore extracted, 1.60

TIMBER, BOARD FEET PER TON

STOPING METHOD	BOARD FEET PER TON O ORE BROKEN	
Horizontal cut and fill	5.4	
Incline cut and fill	3.4	
Square set and fill	14.5	
Shrinkage and fill	3.7	
Glory hole	0.0	
Total stoping		4.7
Drifting	8.5	
Raising	5.6	

TOTAL TIMBER PER TON OF ORE EXTRACTED

Framed timber	0.752
Lagging	3.857
Cribbing	0.870
Bulkheads	0.683
All others	0.640
Total	6.602

HORSEPOWER (TOTAL) PER TON OF ORE DELIVERED TO HOPEWELL BINS

	Kw.-hr.	Horsepower	Cu Ft. Compressed Air
Compressed air (mining)	3.18	256	145
Haulage	1.11	89	
Hoisting	1.14	92	
Pumping	0.24	19	
Ventilation	2.83	228	
Lighting	0.22	18	
Surface plant and miscellaneous	0.49	39	
Total per ton ore	9 21	741	

SAFETY AND WELFARE WORK

The safety department of the Jerome plant is under the direction of one safety engineer, who is responsible to the management through the directing superintendent of the mine. Once a month there is held a meeting of all department heads and here are taken up the general questions of safety as they apply to all the mine. Many matters of safety work in the plant are first turned over to the boss directly in charge and are handled this way to build up in the men the idea that safety work is a part of the work of all employees and not the duty of one man. In order to carry this still farther, the shift bosses are paid a bonus for each 750 shifts without a serious accident and the jigger bosses in the mine are paid a bonus each month for not having a serious accident. First-aid and helmet crews are organized and drilled by this department in order that all possible relief can be had in case of need.

Welfare work is handled by no particular committee or department. Any grievances or work in connection with welfare are handled by the company officials. Recently, a modern apartment house was built for married employees and a rooming house for single men. The steam-shovel department maintains a boarding house and a rooming house. There have been built a swimming pool and a playground for the public and one of the best baseball grounds in the state. Housing conditions are handled by a utilities company, a subsidiary of the United Verde Copper Co.

ACKNOWLEDGMENTS

The author acknowledges the assistance obtained from the paper "Geology and Ore Deposits of the Jerome District," by Louis E. Reber, *Trans.* (1921) **66**, and the paper "Mining Methods and Costs at the United Verde Mine," by H. De Witt Smith and W. H. Sirdevan, *Trans.* (1921) **66**.

Mining Methods at the Homestake

By A. J. M. Ross and R. G. Wayland, Lead, S. D.

(New York Meeting, February, 1925)

The Homestake mine is situated in Whitewood mining district, in the northern Black Hills of South Dakota, in the city of Lead, Lawrence County. The entire property, comprising 557 lode claims with a total area of 3343 acres, is held under United States Patent. The older claims were 300 by 1500 ft. under local regulations, the newer ones 600 by 1500 ft.

The Homestake was discovered April 9, 1876, by Moses and Fred Manuel, and Jake Harney, who were drawn into the northern hills, from the Custer placer district, by news of the discovery of rich placer deposits in Deadwood Gulch. They located a number of additional lode claims and, in a short time, the whole country was covered with locations. The Manuel brothers built an arrastre the following winter and took out $5000.

The next year the Homestake lode was sold to Senator George Hearst and associates, who organized the Homestake Mining Co. Other companies were formed to exploit other groups of claims and at one time five companies were operating on this deposit, the Father DeSmet, the Deadwood-Terra, the Caledonia, the Highland, and the Homestake Mining Co. These companies, however, have been merged into the Homestake Mining Company.

The first stamp mill, known as the "Eighty Mill," commenced operation on July 12, 1878, and production has been nearly continuous since that time.

Geology

The orebodies of the Homestake are found in an intensely folded and altered bed of dolomitic limestone occurring in a series of ancient rocks believed to be of Archean age, belonging to the Keewatin series. The Keewatin series is estimated to be 20,000 ft. thick and is made up essentially of quartz mica schists, garnet schists, quartzites, slates, and some green schists, probably derived from basic igneous rocks. The dolomitic limestone bed has a thickness of 40 to 60 ft. The great width of ore, as much as 400 ft. in some of the stopes, is due to the folding, wherein the bed is repeated. In other cases the bed is stretched and pinched out. Dikes of Tertiary rhyolite cut across the country and, in places, sills overlie the

pre-Cambrian rocks. The ore is found, generally, in the altered dolomite and rarely in green schist, where folding occurs adjacent to the rhyolite dike that cuts through the property. In the outlying districts, the pre-Cambrian rocks are overlaid by flat lying sandstones, quartzites, and limestones of later age. Where it is ore bearing, the altered limestone bed consists of cummingtonite and chlorite with pyrite, arsenopyrite, and pyrrhotite.

Description

Lead is situated among pine clad hills at an elevation of 5000 ft. above sea level; the topography is quite rugged. The climate is similar to that of the northern Rocky Mountain region. Winters are cold but not

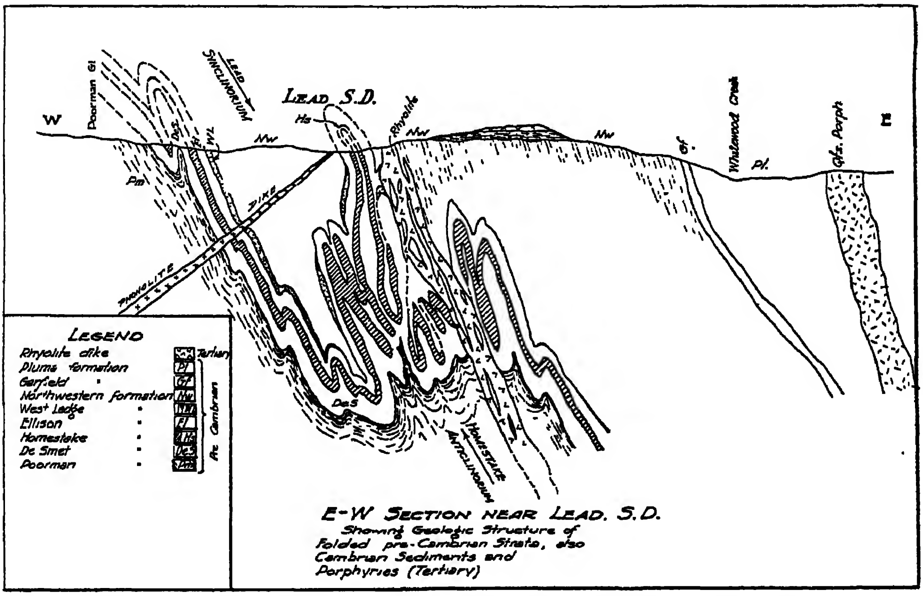


Fig. 1.—Cross-section near Lead, showing the Homestake formation, the ore-bearing bed.

exceptionally severe. Snow may be expected any time from October first to June first. The summer days are warm but the nights are cool. The falls are delightful; the springs, however, are cold and wet. The annual precipitation averages at but 26 in., and varies between 15 and 36 in.

The region is traversed by many streams of excellent water fed by springs. The mine and municipal supply for Lead comes from the watershed of Spearfish Creek, Elk Creek, little Rapid Creek, Whitewood Creek, Deadwood Gulch, and others.

Coal from nearby fields in Wyoming is used for steam power; fuel oil is used in the drill shop and for minor uses.

Two hydro-electric power plants, owned by the company, in the Canyon of Spearfish Creek furnished 35,000,000 kw.-hr. of electric power during 1923.

Timber is supplied by the company's sawmills, one situated 20 miles southeast of Lead, at Nemo, S. D.; the other, 30 miles west at Lavier, Wyo. The timber is excellent western yellow pine and the company's holdings should supply its needs for an indefinite period.

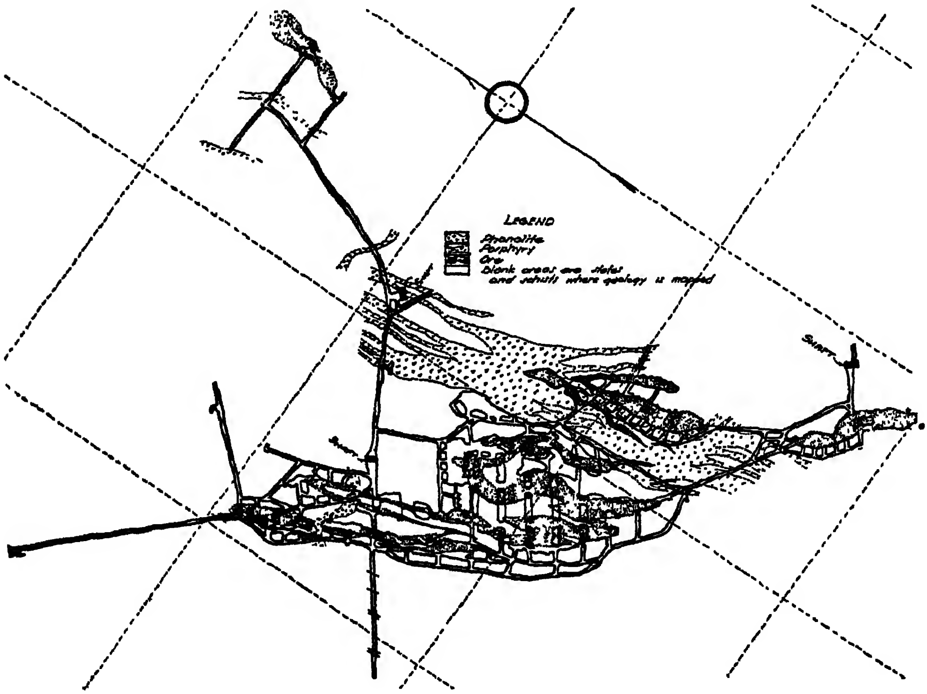


Fig. 2.—Plan of level, showing geology and mine workings.

The labor supply fluctuates. At present, labor is plentiful; 62 per cent. of the men employed are Americans. All can speak and understand English and will compare favorably with the men in any mining camp.

Lead is served by the Chicago, Burlington & Quincy R. R. and the Chicago and Northwestern Ry.

Exploration, Sampling, Etc.

Exploration is carried on by drifting, crosscutting, and diamond drilling. One or two diamond-drill rigs are constantly at work exploring new territory and obtaining information that helps eliminate unprofitable drifting and crosscutting.

The ore is hard and tough. A sampling crew takes channel samples where the development work is in ledge matter. Grab samples are

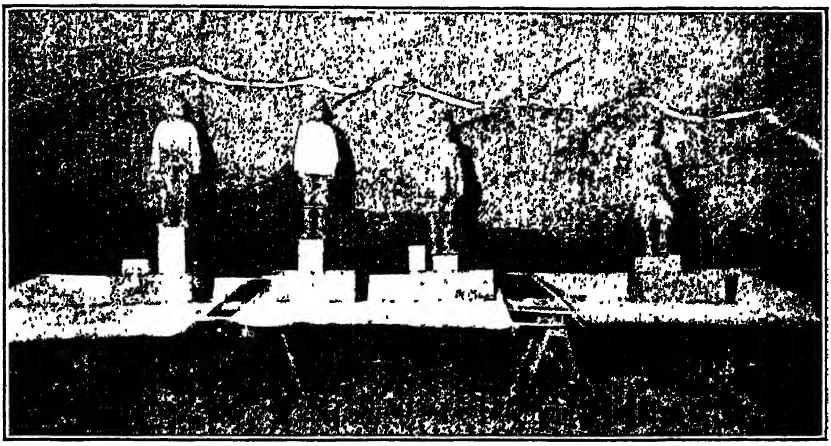


FIG. 3.—SAMPLING THE BACK OF A STOPE ON THE SILL FLOOR.

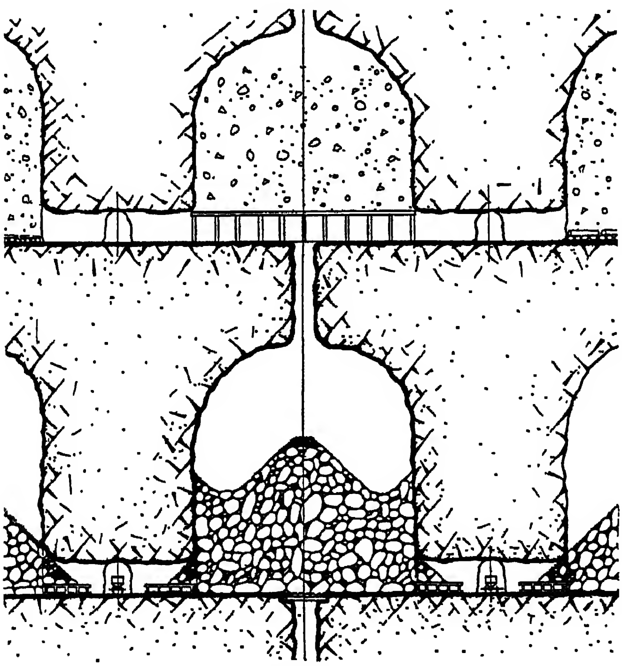


FIG. 4.—LONGITUDINAL SECTION, SHOWING OLD METHOD OF STOPING.

taken from ore cars drawn from caved areas, when there is a question as to whether the contents is ore or waste.

Working and assay plans of each level are kept up to date, together with plans and cross sections of each stope and pillar. The number of cubic feet per ton of ore in place is 10. The method of mining does not contemplate mining losses, but, of course, some are unavoidable.

Results of exploitation indicate that the estimates have been reasonably accurate; however, dilution by waste from caved areas of the old workings has increased the tonnage and reduced the grade.

Principal Mining Methods Used

From the beginning of operations until about 1900, all mining was done by open-cut (glory hole) or square-set methods; there had been no

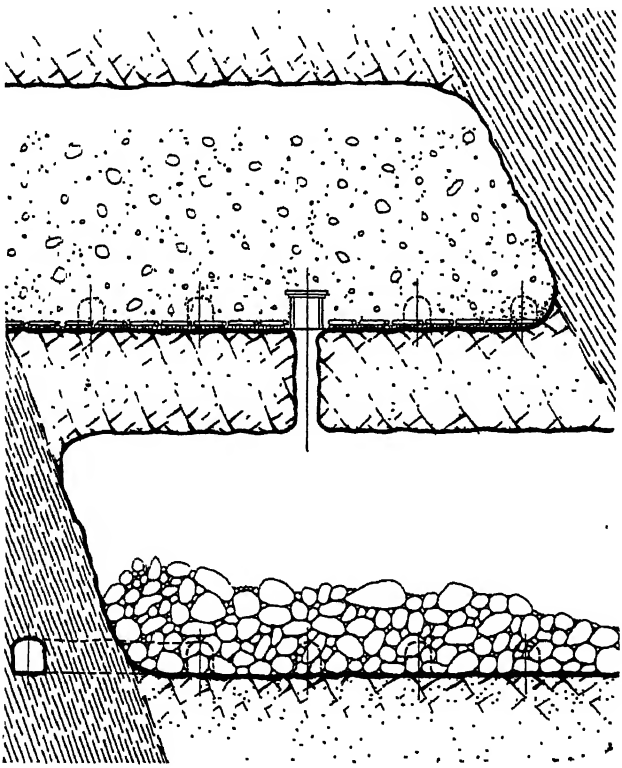


Fig. 5.—Cross-section showing old method of stoping.

system of stopes and pillars. The method was to cut out the sill floor of the ledge to its full size regardless of its length or width. The stopes were then laid out and the space under the pillars filled with solid bulkheads of native pine.

About this time, when the 600-ft. level was being developed, a regular system of stopes and pillars was planned by W. S. O'Brien, then general

mine foreman and B. C. Yates, then mine engineer. This consisted of cutting stopes on the sill floor 60 ft. along the ledge and from wall to wall with 60-ft. pillars between. Stoping is carried to within 25 ft. of the level above.

With this regular system of stopes and pillars, the square-set system was still used in carrying up the stopes. The stopes were filled with waste and the crowns and pillars mined by square setting, after which the sets were waste filled or caved.

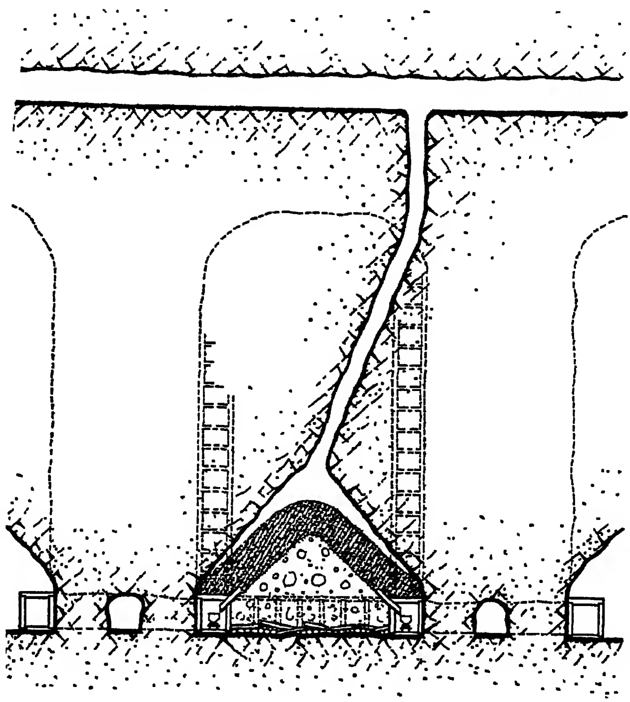


FIG. 6.—LONGITUDINAL SECTION SHOWING PRESENT STOPING METHOD.

The next move was to mine the stopes by the shrinkage system. At first, all the sill-floor square-set timber was "stood" in order to brace the timbers over the trackways, which were laid on each side of the stope parallel to the pillars with cross-tracks connecting. Shortly afterward, this method was changed to standing timber over the track lines only. The trackways were for the purpose of mucking.

The next change was to cut under the pillars, on each side of the stope, about 8 ft. for the trackways, so that the trackway timbers would not have the weight of the broken ore resting upon them. This method avoided breaking the timbers but weakened the pillars somewhat.

After a few years, the size of the pillars was reduced from 60 to 42 ft., with the trackways still undercutting the pillar.

The next change was to drive a 7 by 7 ft. crosscut through the center of the pillar, from wall to wall, and drift from it at right angles every 30 ft. to the stope on each side. Through these drifts, the swell in the shrinkage stopes was shoveled into cars. This method eliminated the use of timber in shrinkage stopes and gave a crosscut to work from during the mining of pillar and crown. It has the disadvantage of weakening the pillar.

Until 1917, all the ore from the shrinkage stopes had been shoveled into cars on the sill floor, for the reason, mainly, that the ore broke into

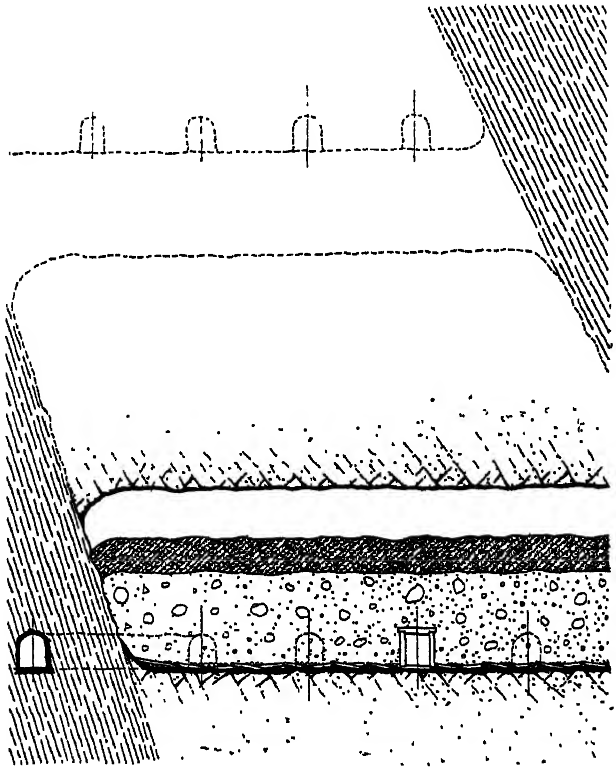


FIG. 7.—CROSS-SECTION SHOWING PRESENT STOPING METHOD.

such large pieces that it required block-holing before it could be loaded into cars. In some cases, a man could block-hole enough ore to keep three shovelers busy and in others only two. This difficulty was overcome by block-holing the ore on top of the pile immediately after it was shot from the solid, and chutes were built that enabled most of the ore to be drawn into cars. The broken ore on the sill floor that cannot be drawn is still shoveled from the sill floor. In the past, this ore has been shoveled last; two years ago this was changed. We now cut out the sill floors and arch the back in the shape of the pile of ore that would be left

after the chutes have drawn all that is possible. This broken ore is now shoveled out and the space filled with waste with the exception of two chute lines, one on each side of the stope parallel to the pillars. In the

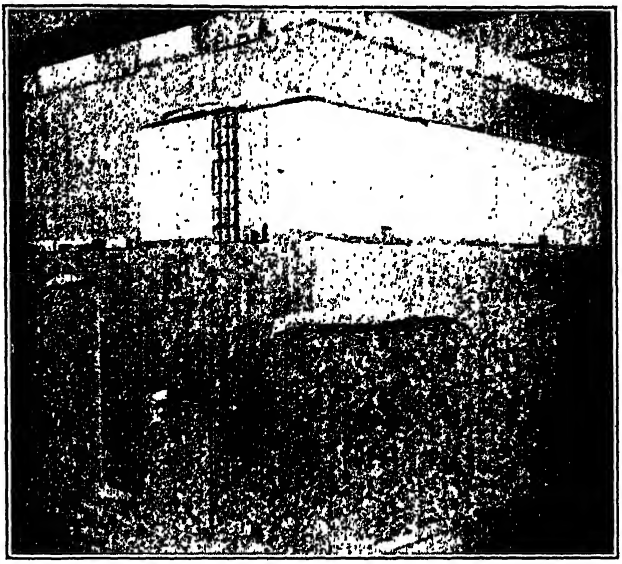


FIG. 8.—HOMESTAKE MINE MODEL.

FIG. 9.—CUTTING OUT THE SILL FLOOR.

smaller ledges, one chute line is placed on the foot wall and waste is trammed to fill the hanging-wall side up to the angle of repose.

Until about 25 years ago, no backfilling had been done in the mine. In large orebodies, like the Homestake, if backfilling is not done, extensive

caving or subsidence may be expected. As a result, the main ledge of the mine gradually crushed from the 500-ft. level to the surface. In this area was left an immense tonnage of ore that could not be mined at a profit by the square-set method. For a number of years a small part of this ore was secured by what is locally known as "draw holes," or raises tapping the caved area. Early in 1916, a more determined effort was made to extract this ore. The importance of this ore may be judged by the fact that for the past 7 years 50 per cent. or more of the total tonnage mined has been by draw-hole mining.

The early draw hole was a nine-post raise the necessary height in the solid rock alongside the caved zone. The raise was then broken through

Fig. 10.—Shrinkage stoping, before days of block-holing on top of pile.

into the caved zone and the rock drawn over a grizzly placed on the top floor. As ore is drawn through the chutes on the sill floor, the chute drawer sorts the waste from the ore.

The present method is to drive a 6 by 6 ft. raise in the solid for the grizzly raise and another of the same size about 20 ft. away for a manway. This manway is divided into two compartments, one for the ladders and the other for the purpose of dropping the timber that is drawn from the old stopes. The tops of two raises are connected by a manway drift and a grizzly is installed. The first cost of the two raises is more but the maintenance much less than for the nine-post raise, and the time lost during stoppages for repairs is a great deal less.

From the surface to the 1100-ft. level, all levels are 100 ft. apart. From the 1100-ft. level down, they are 150-ft. apart. In opening a level a crosscut or drift is driven from the shaft to reach the orebody in the shortest distance. After the ore has been reached, a foot-wall drift is usually started in each direction and the foot wall is followed to the limits of the ore. In the past, the orebody has been developed by driving a crosscut through the center of each pillar to the hanging wall and for a distance of usually 30 ft. into both hanging wall, and foot wall. These crosscuts are then connected at the ends, giving foot wall and hanging wall headers parallel in the orebody and about 30 ft. from the ore. In the future, the crosscuts will probably be driven through the

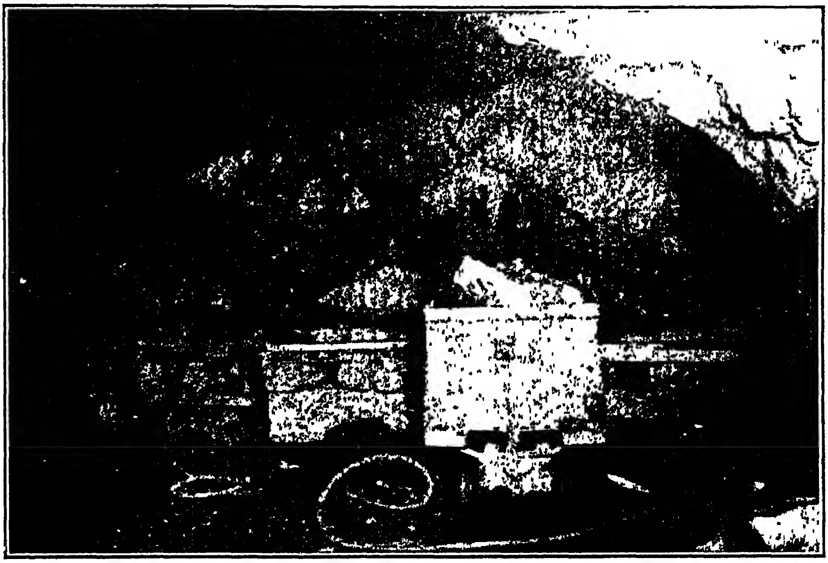


FIG. 11.—SHOVELING SHRINKAGE STOPE ROCK FROM A TIMBER PLATFORM.

center of the stopes, leaving the pillars solid and making it possible to reduce their thickness.

As previously stated, the ore occurs in an intensely folded and altered bed of pre-Cambrian dolomitic limestone. The orebodies are very irregular in shape and vary greatly in size. The main ledge may be ½ mile long with widths up to 400 ft. due to repetition of the cummingtonite bed. Other orebodies on the same bed may be less than 20 ft. wide and not 100 ft. long. The noses of the anticlines and troughs of the synclines plunge at low angles and, with the erratic folding, cause marked differences in shape and size of orebodies from level to level. Each stope presents a problem peculiar to itself.

The ore is hard and tough and stands wonderfully well, as does the wall rock. The average tenor of ore hoisted is $4 per ton, mainly in

gold but with a small amount of silver. Water has little effect on the clean ore but makes a sticky mess of the draw-hole rock on account of the consistency of some of the waste. Rock temperatures are low and, where ventilation is provided, have no effect on operations. The Ellison shaft, now the main hoisting shaft, is a five-compartment shaft 12 by 20 ft. outside of timbers. There is one cage compartment 5 by 10 ft., two skip compartments 5 ft. 6 in. by 5 ft. 8 in., a pipe and ladderway compartment, and a compartment for cage counterweight and electric

FIG. 12.—DRAW HOLE AND GRIZZLY.

cables. It is constructed of timber. The bottom level is the 2300-ft. level.

The B & M shaft is a five-compartment shaft consisting of two cage compartments, two skip compartments, and a pipe and ladderway. This shaft is of timber construction and serves all levels down to the 1500-ft. level.

The Old Brig shaft is a small shaft consisting of two cage compartments and a pipe compartment; the construction is wood. It reaches the 800-ft. level.

The B & M No. 2 shaft is now being sunk to the 900-ft. level to replace the top section of the B & M, which is threatened by ground movement.

Underground Development

Double-track drifts are 7 by 12 ft. Single-track drifts are 7 by 7 ft. Water is carried in a ditch at one side of the track. Plain board chutes are used.

Mining

All drilling in shrinkage stopes is done with machines mounted on posts so that nearly horizontal holes can be drilled in the back of the

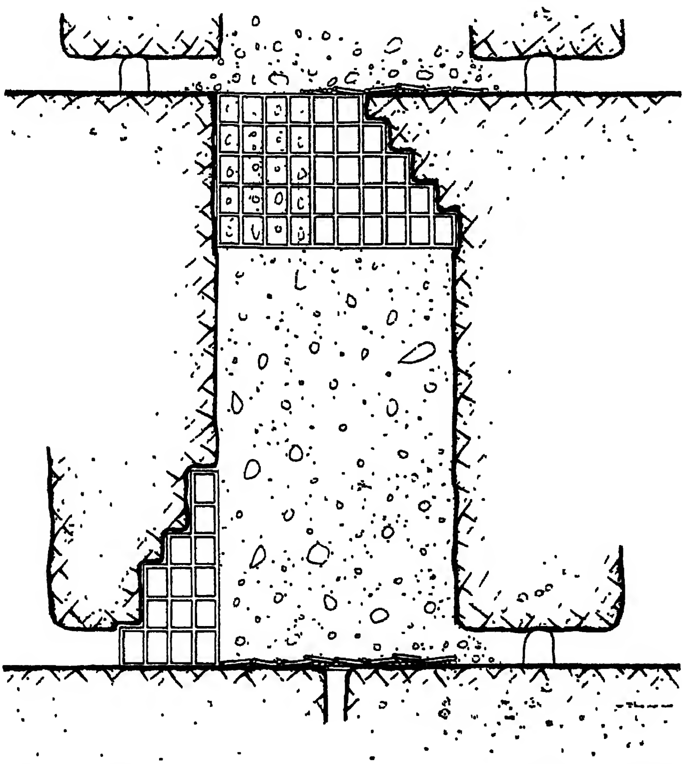


Fig. 13.—Method of recovering crowns and pillars after stopes are waste filled.

stope. Holes "looking up" too much leave a dangerous roof. Two types of drill are used, the Cleveland No. 5D and the Ingersoll-Rand No. 248. These machines are also used for drifting. For raising, the Ingersoll-Rand CCW11 and Cleveland No. 44SW stopers are used. Cleveland No. 44TW rotators are used for block-holing. All these machines are fitted with anvil-block chucks; 1-in. quarter-octagon hollow drill steel is used for all machines. Double-taper cross bits are used for drifting, raising,

and stoping while a rose bit has been found most satisfactory for block-holing. Straight shanks are used.

No standard round is used as nearly all of this work is done by contract, and the contractors place the holes in the manner best suited to break the ground they are driving through.

For all mining and block-holing, 40 per cent. gelatin, 1⅛ by 8 in., is used. A small amount of Monobel is used to blast timbers.

After all the ore has been drawn from the shrinkage stopes, the stopes are filled with waste. Some of this waste is obtained from development

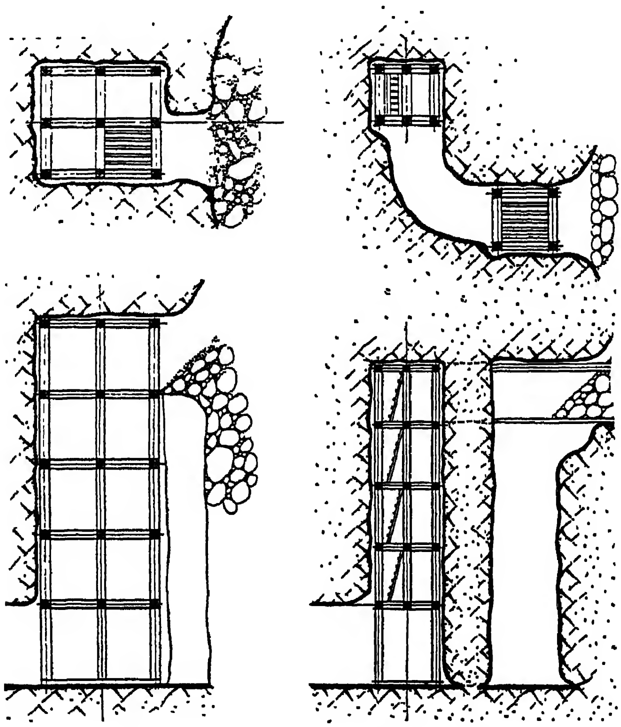


FIG. 14.—TYPES OF DRAW HOLES.

work, some from draw holes that are placed under the crushed area for that purpose, and the rest is sorted from the product of the raises that are drawing ore from the crushed area. Most of the waste is trammed to a centrally located transfer raise, where it is dumped. It can be drawn on any level. A short inclined raise is driven from each level to tap the main raise. Waste can be dumped into the waste raise on any level down to the 900-ft.

Timbering

Timber is used in the following places underground: Shafts, shaft stations, drifts and crosscuts through filled stopes and heavy ground

Square sets are used when removing pillars and crowns. The shaft timbers are 12 by 12 in. and may be yellow pine or Douglas fir. For lagging 2 by 12-in. plank is used. Drift sets are pine and are either hewed 11-in. timber or sawed 12 by 12 in. The lagging is either 6 in., slabbed off on two sides, or two layers of 3-in. flat slabs. Square sets are pine and are framed at the mill at Nemo; 6-in. lagging is used.

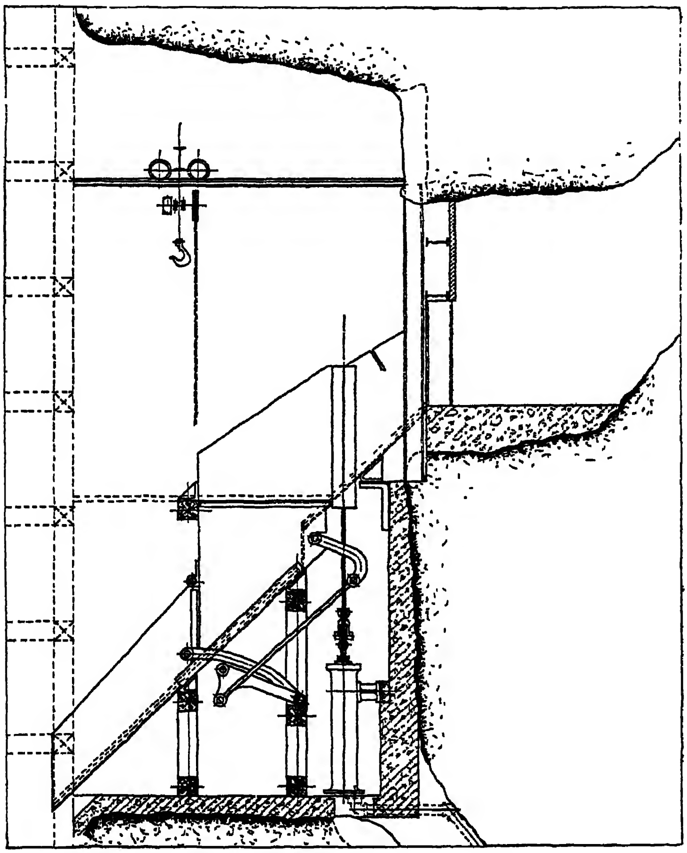


Fig. 15.—Skip loader.

Tramming

Haulage is by high-pressure compressed-air locomotives, of which there are 29 in the mine. Twenty are 5-ton locomotives having 6 by 10-in. cylinders and a tank capacity of 40 cu. ft. of air charged at 900 lb. Nine are 3½-ton weight with 5 by 8-in. cylinders and a tank capacity of 20½ cu. ft. A reducing valve on each locomotive admits the air to the cylinders at 160 lb. High-pressure air is piped throughout the active levels and charging stations are placed where necessary.

Cars are of 20 cu. ft. capacity; the older types are end-dump and the newer, side-dump gable-bottom cars. The mine track gage is 18 in. and 40-lb. rails are used on the main haulageways; 25-lb. rails are used on development drifts.

Underground Storage and Dumping

There are two main hoisting shafts, the B & M shaft, which has been operating as at present equipped since March 1917, and is now about to

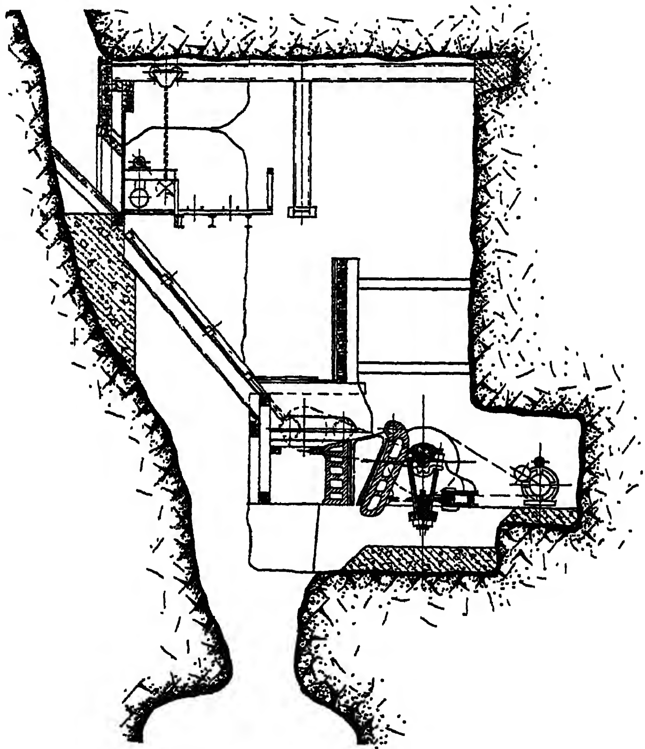


Fig. 16.—Underground crusher station.

go out of service, and the Ellison shaft, which has been remodelled and has been operating about one year.

There are three loading stations at the B & M shaft equipped with measuring pockets. These stations are located 35 ft. below the 800-ft., the 1100-ft., and the 1550-ft. levels. A small storage bin is located immediately above each loading station and transfer raises from the remaining levels afford ample storage capacity.

At the Ellison shaft, the primary crushing is done underground on three levels; the 800-ft., the 1400-ft., and the 2000-ft. level. The crushers are 36 by 48-in. jaw crushers fed by apron feeders. Below each crusher station is a 1500-ton ore bin and a loading pocket. Fig. 16 shows

the arrangement of the crusher station and Fig. 15 the pocket. Transfer raises from the crusher stations to the remaining levels, together with the ore bins, afford a storage capacity of 12,000 tons, of which 4500 tons is crushed to 4½-in. size. The loading pockets are from 150 to 200 ft. below the levels on which the crusher stations are situated.

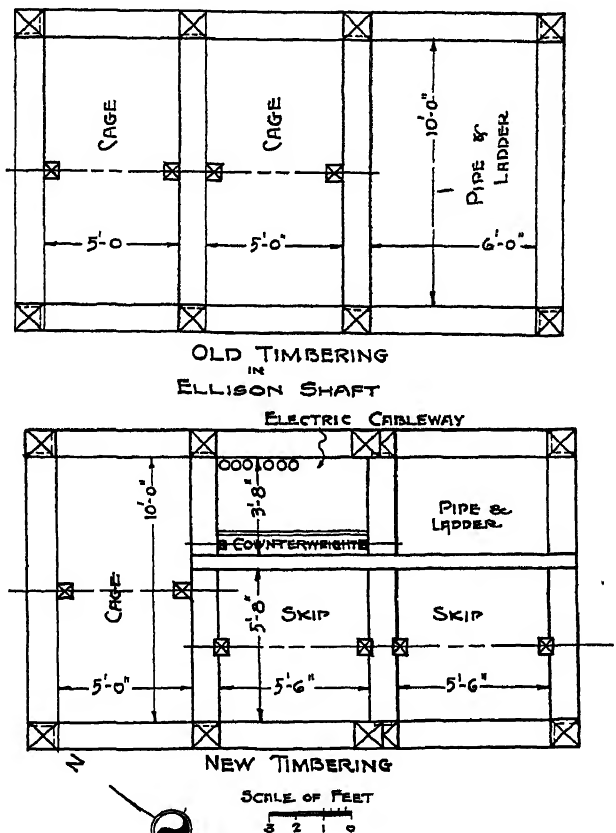


FIG. 17.—ELLISON SHAFT TIMBERING.

Hoisting

At the B & M shaft, the ore is hoisted by a Nordberg, four-cylinder, A type, compound, condensing, Corliss valve, first-motion, steam hoist. The 8 by ⅝ in. flat rope is wound on two reels, each having a small diameter of 6 ft. 4 in., and each mounted on a separate shaft. The distance center to center of ropes is 18 in. The two high-pressure cylinders are 28 in. in diameter, the two low-pressure 52 in., with a 42-in. stroke. The estimated horsepower is 2500, and the hoist is designed to raise 6 tons of ore from a depth of 3000 ft. A "Lilly" controller provides against

overspeed and overwind. The hoist has operated since 1917. Hoisting is in counterbalance.

The man hoist at the B & M shaft is a Fraser & Chalmers, double-cylinder, slide-valve, first-motion hoist, with cylinders 18 and 18 in. and 60-in. stroke. This hoist was brought by bull-team from Fort Pierre on the Missouri River, and installed in 1879. It winds a 4 by 5/8 in. flat rope on reels.

The winding part of the Ellison ore hoist is a 36,000-lb. rope pull, 2000-ft. per min. rope speed, double-drum, electric hoist, with plate-steel drums 10 ft. in diameter by 7 ft. 2 in. face, structural-steel post brakes,

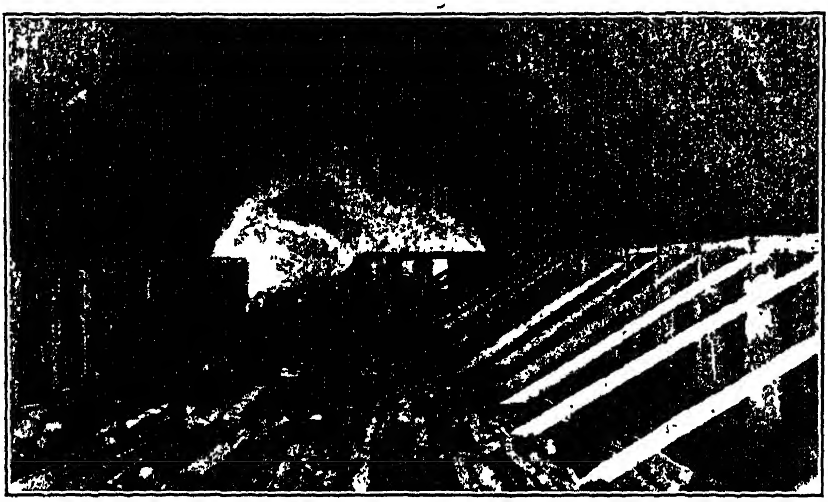


FIG. 18.—SILL FLOOR, NEW SYSTEM, SHOWING TRACK LINES AND TIMBER MAT READY FOR WASTE FILLING.

plate clutches, oil-operated auxiliary engines, and complete safety features. The rope is 1½ in. diameter, Lang lay, seal construction, extra plow steel. The hoist is direct-connected to a 1400 hp., 600-volt, direct-current motor, having a full load speed of 63½ r. p. m. The current for this motor is supplied by a flywheel motor-generator set, and the control is Ilgner-Ward-Leonard. The speed of the flywheel set is 600 r. p. m. The weight of ore hoisted is 7 tons per skip load. The hoist is designed to operate from a depth of 3230 ft. Hoisting is in counterbalance. Allis-Chalmers built the winding part and the General Electric Co. the electrical.

The Ellison man hoist was built by the Union Iron Works in 1896. It is a double-acting, poppet-valve, steam hoist with cylinders 26 in. in diameter by 72-in. stroke. Two reels hold enough 8 by 5/8 in. flat rope to reach the 2300-ft. level. The hoist has been in operation since 1902.

At the Old Brig shaft, a small motor-driven double-reduction geared hoist is raising about 500 tons per day in cars on cages; this will soon be discontinued. A 1-in. round rope is used.

Pumping

The average amount of water pumped from the mine is 400 gal. per min. This is accomplished in lifts of 300 ft. by motor-driven, three-stage, centrifugal pumps of 1000 gal. capacity placed two on each main pumping level where a suitable pump room and sump are provided. There are two 1000-gal. pumps on each of the following levels at the B & M

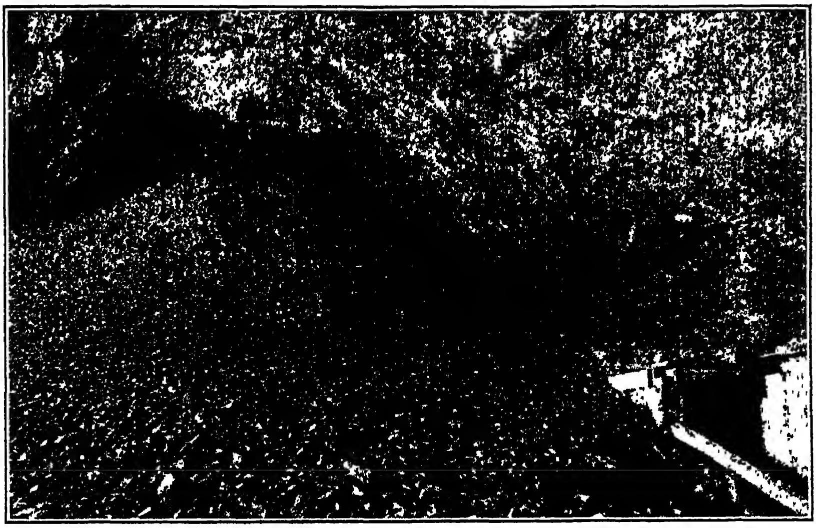


Fig. 19.—Filling sill floor preparatory to back stoping on new system.

shaft: 500-ft., 800-ft., 1100-ft., and 1400-ft. At the Ellison shaft, there is one 500-gal. pump on the 1700-ft. level and one on the 2000-ft. level.

Air Compressors

High-pressure air for haulage is furnished by one Ingersoll-Rand Pre-4, four-stage, 1000-lb., compressor, driven by synchronous motor at 128.2 r. p. m. The cylinder sizes are 24 in., 14 in., 11½ in., and 6 in. by 24-in. stroke. As a stand-by unit, there is an Ingersoll-Sergeant four-stage compressor, formerly steam-driven but now motor-driven by synchronous motor through a rope drive. The air-cylinder diameters are 37¼ in., 20¼ in., 12½ in., and 6 in. by 48-in. stroke. The speed is 68 r. p. m. Another auxiliary high-pressure compressor is steam-driven and its air cylinders are 24¼ in., 14½ in., 9¼ in., and 4½ in., by 42-in. stroke.

The low-pressure, 100-lb., air for rock drills is furnished by four machines. One of these is a Nordberg compressor of 3200 cu. ft. capacity

with cylinders 19½ in. and 32½ in. diameter by 44-in. stroke, driven by synchronous motor through a rope drive. There are two Ingersoll-Rand Pre-2 compressors with cylinders 39 and 23 in. diameter and 27-in. stroke, direct-connected to synchronous motors. The speed is 150 r. p. m. and capacity 4500 cu. ft. The fourth is an Ingersoll-Rand Pre-2 with 40 and 25½-in. cylinders and 30-in. stroke, direct connected to a synchronous motor and driven at a speed of 138½ r. p. m.

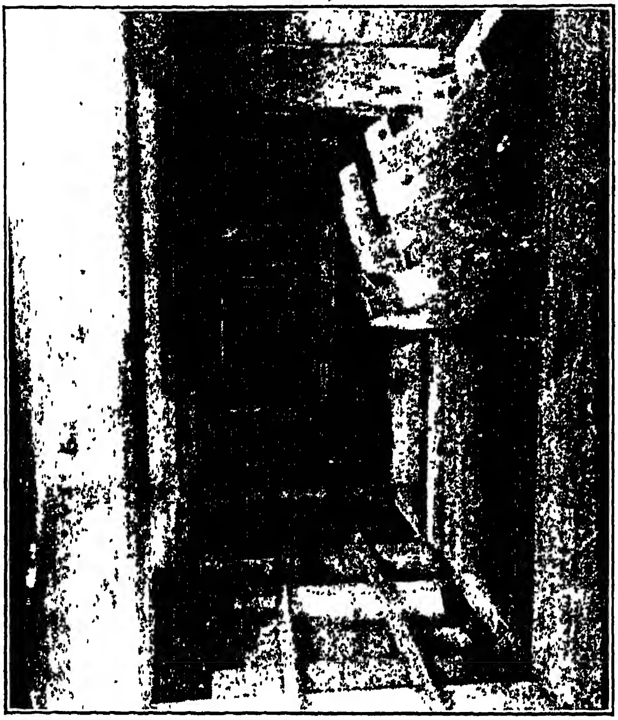


FIG. 20.—TRACK LINE WITH CHUTES, NEW SYSTEM.

Ventilation

Artificial ventilation is produced by a 250,000 cu. ft. per min. exhaust fan placed at the collar of an inclined air shaft. One 60,000 cu. ft. auxiliary fan is operating on the 1400-ft. level at the B & M shaft and a duplicate on the 1800-ft. level at the Ellison. Small fans are used when necessary to ventilate isolated working places.

Lighting

Electric light is used in shaft stations and throughout the levels. Carbide lamps are used for individual lighting.

Telephones and Signals

At each shaft, each level is connected by telephone with the company's central. Each loading pocket is connected directly by telephone with the hoist platform.

The haulage is controlled by a block-signal system using red and green lights. Electric signals are given in the shafts, both for ore and men. It is so arranged that the level from which the signal is given is indicated to the hoist engineer.

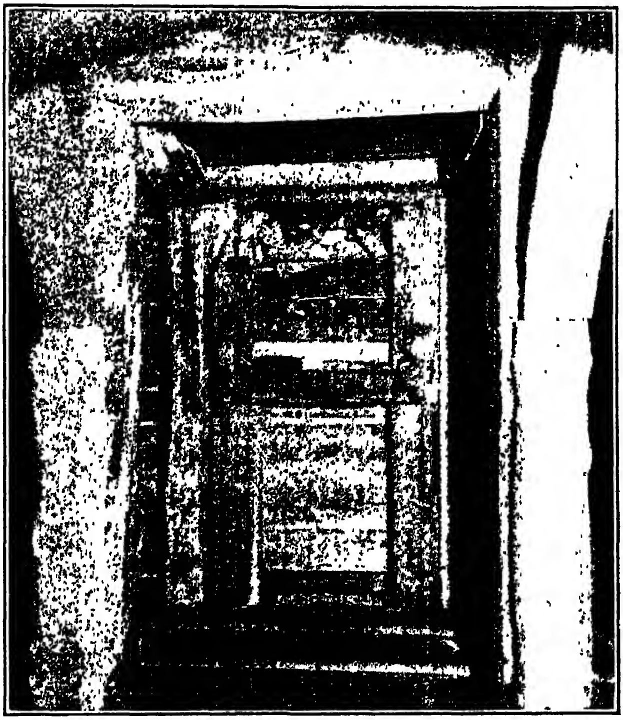


FIG. 21.—HOMESTAKE BOARD CHUTE.

Records of Unit Production

The tonnage mined in 1923 was 1,652,337 short tons.

Contracts are let on all work done in the mine. Nearly all development work is done by contract, and is paid for by the foot. About 50 per cent. of the square-set mining is by contract and is paid for on tonnage basis. The greater part of shoveling ore from shrinkage stopes is done by contract at so much per one ton car, each contractor doing his own block holing. In each case, the contractor buys his powder, fuse, and caps, which are furnished at cost.

Tons produced per man-hour for miners is 2.5; man-hours per ton is 0.4.

Tons produced per man-hour for all underground labor is 0.75; man-hours per ton is 1.33.

Tons produced per man-hour for surface labor is 5.6; man-hours per ton is 0.18.

Tons produced per man-hour for total organization is 0.66; man-hours per ton is 1.5.

Mine labor is 88 per cent. and surface labor 12 per cent. for operations that bring the ore to the shaft head-frame pocket.

Mine labor turnover is about 75 per cent. of the number on the mine payroll per year.

Total direct mine labor cost is 62 per cent. of the total cost of mining.

The following figures carry the mine costs up to the point of dumping the ore into the shaft head-frame pockets:

About 0.6 lb. dynamite are used per ton of ore mined.

Mine timber used per ton of ore hoisted is 1.0 ft. B. M., lagging and square slabs 0.7 ft. B. M., and lumber 0.3 ft. B. M.

The following table shows the distribution of power to underground operations:

	Horsepower-hours per Ton Ore Hoisted
Rock drilling, drill sharpening, blowing, etc.; compressed air 100 lb.	5.13
Haulage, underground and surface, compressed air 100 lb.	1.76
Ore hoisting	2.79
Man hoisting and errand work	0.25
Mine pumping	0.96
Mine ventilation	1.84
Mine lighting	0.33
Total	13.06

Cost of direct supplies used but not taken into account above is 18 per cent. of total mining cost.

Total cost of all direct supplies, expressed in percentage of total cost of mining, is 30 per cent.

The ore is milled on the property and bullion is shipped.

Safety and Welfare Work

Safety work is taken care of by a safety engineer working in conjunction with a central safety committee, composed of department heads, and a workmens' committee composed of representatives elected from among the employees. The central committee meets each month and the workmens' committee every other month with the central committee. The committees report on conditions, discuss accidents and their prevention, and pass on suggestions.

First-aid and mine-rescue training also come under the supervision of the safety engineer.

The company maintains a hospital for its employees and their families. The hospital staff consists of five physicians and the necessary number of nurses. This service is rendered, both in cases of accident and sickness, without cost to the employee.

A recreation building, or employees' clubhouse, is maintained for the benefit of the employee and the people of Lead. It contains a free library, billiard and pool tables, bowling alleys, swimming pool, and theater. A charge is made only in the theater.

An Employees Aid Fund is maintained, whereby the employees mutually insure themselves against sickness and death. One dollar and fifty cents per month is collected from each man. The sick benefits are $1 per day for the first six months and $1.50 for the next three; $800 is paid in case of death, except accident, which is provided for by compensation, or in case of suicide. The fund is administered by a Board of Directors, consisting of employees.

DISCUSSION

Benjamin F. Tillson,* Franklin, N. J.—It requires a good deal of timber to operate, especially the caving systems. An important function in mine operations now seems to be forestry and to operate a timber tract in order to insure a supply of timber. By depleting the timber resources of your section, you only increase your costs later when timber must be brought in from the Pacific Coast or the East.

Our experience has been that any square-set method of timbering is the most expensive. Timbering for stoping systems, top slicing if you will, although it uses a large volume of time does not require as expensive timbering as the square-set method; also it does not require as long a life for the timber, so that such wood as birch and maple, which have strength but not long life under the fungus conditions in the mine, may be used. All the drifts in the orebody are thoroughly timbered, and it has required a lot of timber. If we had it to do over, we would spend a lot more money in the development of rock drifts; as a matter of fact, we must ultimately come to that anyway.

R. M. Raymond,† New York, N. Y.—It is a very important point and one worth considering—that in laying out a mine the levels should be run at a distance apart to leave a good rock arch over the drifts where possible, thereby also saving the proportionate amount of timbering as stoping rises to the upper level. A large amount of timber can be saved by developing in rock and rock drifts and crosscuts rather than

* Mining Engineer, New Jersey Zinc Co.

† Professor of Mining Engineering, Columbia University.

putting in timber. It is much more convenient and the Homestake must have had considerable delay and expense in the retimbering as the timber became decayed or crushed, in addition to delay in the tramming.

I. H. Barkdoll,* Globe, Ariz.—One cannot compare the porphyry mines of the Southwest with either the mines of Butte or the Homestake; although the Homestake may have peculiarities applicable, the rock is much harder. The amount of timber used by us has always been quite an amount, in some cases as high as 20 cents per ton. This is gradually being decreased, partly by caving the blocks in shorter lifts and as the weight gets excessive in the drifts, dropping below for new ones. A very small amount of timber is used above the haulage drifts.

Arthur Notman, New York, N. Y—When they could determine in advance how the orebody lay they kept in the solid limestone as far as possible. The expense of timbering in mines where the limestone has been subjected to oxidizing action is very great. They are more or less limited to square-setting or modified top slicing. As Mr. Tillson has pointed out, you cannot get away from the expense of timber in square-set methods. As I recall, the figures for timber per ton mined in post-war years up to 1923 (practically the whole production was coming from square-cut or top slicing) amounted to about 16 ft. board measure per ton mined in stoping alone. Adding the timber used in development work, made a total of about 20 ft.

Benjamin F. Tillson.—At Franklin, the days of shrinkage stoping have largely passed. We have experimented with steel and a careful study has shown that it could be employed with economy in shrinkage stoping; that steel could be used in place of timber without increasing the cost. So far as the replacement of timber is concerned, in shrinkage stoping which was later filled, we found it better to use steel as we could recover the steel for further use.

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*Superintendent of Mines, Old Dominion Copper Min. & Smelt. Co.

Mining Methods in Zaruma District, Ecuador

By Rudolph Emmel,* Guayaquil, Ecuador

(New York Meeting, February, 1925)

The mines operated by the South American Development Co. are located in the Zaruma mining district of southwestern Ecuador. They are near the old mining town of Zaruma, which is the only important city in the canton of the same name. The district is situated in the upper end of a valley lying between two spurs of the Cordillera, one of which may be considered the Coast Range and the other an intermediate range. The mining camp proper, known as Portovelo, and the plant are 2.4 km. south of Zaruma, on the north bank of the Amarillo River, a tributary of the Tumbes River, which flows southwestward through Peru to the Pacific.

Portovelo is difficult of access; it is reached from Guayaquil by means of river steamers and muleback. Embarking at Guayaquil, the route is down the Guayas River, across the Jambeli Canal, and up the Santa Rosa River to Santa Rosa, a distance of 177 km. From Santa Rosa, the road or trail follows the Santa Rosa River, then across the summit of the Coast Range into Portovelo, a distance of 74 km.—a two-day trip on muleback.

The mining property of the South American Development Co. comprises 297 lode claims and 142 placer claims. The combined lode claims cover an area, roughly rectangular in shape, 9000 m. long from north to south by 4000 m. wide; the northern edge of the rectangle is about 3000 m. north of the central plaza of Zaruma. Mineral lands in Ecuador are held by virtue of an annual tax per claim paid to the central government, no surface rights being included.

History

The Zaruma mine has been worked by white men since 1549, probably before that it was worked by Indians. The followers of Pizarro, led by traces of gold in the sands of the Tumbes River, followed that stream to its source, worked the gold veins that they found and established the city of Zaruma. Among the mines worked by them were th Sesmo, Leonora, Viscaya, and Mina Grande. Only the soft ores at the outcrop were

* Mine Superintendent, South American Development Co.

mined, making many shallow openings, none being of any considerable vertical extent. Traces of these old superficial workings have been important in outlining recent development.

Foreign capital became interested in the district in 1875, when a Chilean company was formed to work several of the old mines. Its efforts met with little success, as did the attempts of a few small, contemporary, native companies; in 1880, an English company, the Great Zaruma Mining Co., was organized to take over the properties. Later this was reorganized as the Zaruma Gold Mining Co. and carried on operations for several years; it spent considerable money on road construction and the building of a 20-stamp mill. Sporadic efforts to open up the Portovelo mine were made, some stoping was done, and some bullion shipped. Later, the South American Development Co. acquired the rights and property of the English company, together with those of other smaller companies; it has since increased its holdings through purchases and denouncements. Since 1900, there has been continuous exploitation of the mines of the Zaruma mining district by practical modern methods.

General

At present, the mines are operated in three units, known as the Portovelo or shaft mine, the Soroche, and the Jorupe. In addition, considerable work has been done in opening and prospecting other old mines of the district.

The mines are located 610–914 m. above sea level. The district is rugged, being cut by many small ridges and streams. The climate is tropical to subtropical, the temperature ranging between 82° F., at midday, to 62° F., at night. The dry season extends from about the first of June to the last of December, with occasional showers, and the wet season from December to June, with late afternoon showers; the average annual precipitation is about 70 inches.

Power is obtained from the Amarillo River through two canals, the water being used on turbines or Pelton wheels and the power used directly or converted into electricity to be transmitted and used in various operations.

For timber, the mine is dependent on native supplies; that within a radius of 5 miles has already been used. Native contractors supply all timber, dragging it in with mules during the dry season. The maximum size of round timber is 12 in.; and of squared 9 in. in lengths up to 10 ft. Timber is not seasoned, for it appears to last better when placed underground in a green condition. The hard, heavy woods, sanon and amarillo, are used extensively for timbering and tarapo in 4-in. poles for lagging. While the woods are very hard, they are short-fibered and do not withstand blasting well.

The mine freight from the coast, averaging about 750 tons yearly, is brought in on muleback during the dry season, by contract. The cargoes are arranged in loads of about 200 lb. per mule. Some heavy pieces are slung between two mules while a string of mules is used to carry cables, each carrying sufficient coils to make a load of 200 lb.; exceptionally heavy pieces are lashed to bamboo poles and carried by relays of men. The company has improved the trail in many places, cutting out fords and paving muddy places. The movement of freight from the coast costs about $35 per ton, hence high-grade material is used throughout the operations—high-strength dynamite, high-strength cyanide, special alloy steels, etc. Supplies are ordered through the New York office and purchases are made to the best advantage either in the United States or Europe. Delivery is made at Puerto Bolivar, where it is placed in river steamers that carry it to Santa Rosa, the lower terminal of the mule trail. The long time between ordering and delivery at mine necessitates placing the orders nearly a year in advance.

The mine labor is nearly all native Ecuadorian (a mixture of Spanish, negro and Indian); a few Columbians, Peruvians, and negroes also are on the pay roll. Bonuses are paid to men working 20 shifts or more per month, but the many religious holidays interfere materially with steady work. Nearly all work is done by contract. There is one pay day per month, but the men may draw from day to day the greater part of what is due them.

The Ecuadorian unit of currency is the sucre, with a par value of $0.4878. Recently it has fluctuated through wide ranges, therefore it is practically impossible to give the operating costs in dollars.

Geology

Granites and syenites, connected with gneisses and crystalline schists of Archean age, are the dominant rocks of the eastern range or main Cordillera, while in the Coast Range and inter-Andean country greenstones and porphyries are found in connection with Cretaceous formations.

The gold-quartz veins worked by the South American Development Co. occur in a belt of greenstone. The dominating structural feature (shown in Fig. 1 by the heavy line) is a major fault zone, known locally as the Abundáncia fault, that has been traced several kilometers on the surface. Its general strike is N. 3° W., with an average dip of 65 to 68° to the east. Underground workings have opened up this fault for about 2000 m. along the strike. It is always strong and well defined, with a heavy gouge indicating a great deal of movement, and contains more orebodies than any of the other known fissures. Several lesser fractures making away from this fault at small angles are ore carriers; these are known as Cantabria, Portovelo, Soroche, Twenty-six, Nudo, and Quebrada veins. Tamayo and Jorupe veins are outlying fractures;

their relation to the Abundancia fault has not been proved but they are commercially valuable. The San Guillermo vein, worked in the Soroche mine, is without doubt a development on the Abundancia fault proper and should not have a separate name. The Agua Dulce vein, which is in the development stage and on which some ore has already been found, may be an extension of the Cantabria system of mineralization. What is known as the Portovelo vein, in the southern part of the property, is really an Abundancia fault, and should be known as such.

The rock in either wall of the fault is dacite. Near the veins it is considerably altered. The width of the altered dacite at the southern end is about 15 m., while at the northern end where there are more branch veins, the zone is 200 m. wide.

About 2.7 km. from the Soroche mine is a sharp conical peak of rhyolite; it is quite possible that the veins of the Zaruma district are genetically related to this intrusion of rhyolite.

The veins as developed along the Abundancia and Portovelo faults are composed of an intergrowth of quartz and massive calcite, with subordinate amounts of iron and copper sulfides, sphalerite, and galena. Considerable gouge is present on the foot wall, indicating much movement; and in some of the stopes, pronounced brecciation and recementing of the breccia is distinctly evident, showing at least two stages of mineralization. Some of the branch veins have different characteristics, indicating different periods of mineralization. The Twenty-six vein, in the upper levels, is practically pure quartz; but, with depth, it approaches the Abundancia type. Soroche and Tamayo veins consist of ribs of hard quartz, alternating with soft sugary quartz and without calcite. Cantabria, Nudo, and Agua Dulce veins, while having quartz and calcite in normal quantity, carry an excess of sulfides. Jorupe vein carries little calcite but large amounts of sulfides. It appears as though the gold-bearing solutions were introduced into the various fissures through the Abundancia fault, but penetrated only a certain distance from it. In all veins where high-grade ore is found, tetrahedrite is present.

The major fault, the Abundancia, represents the oldest of a series of faults; all others are of minor importance. The other veins never break through the Abundancia fault, but have a tendency to flatten against it and parallel it before they pinch out.

Vein Descriptions

The veins as opened by underground workings on A level, or projected on to that level from others where they have been opened up, are shown in Fig. 1.

Abundancia fault, while easily traceable throughout the lateral extent of the underground workings, is not always marked by the pres-

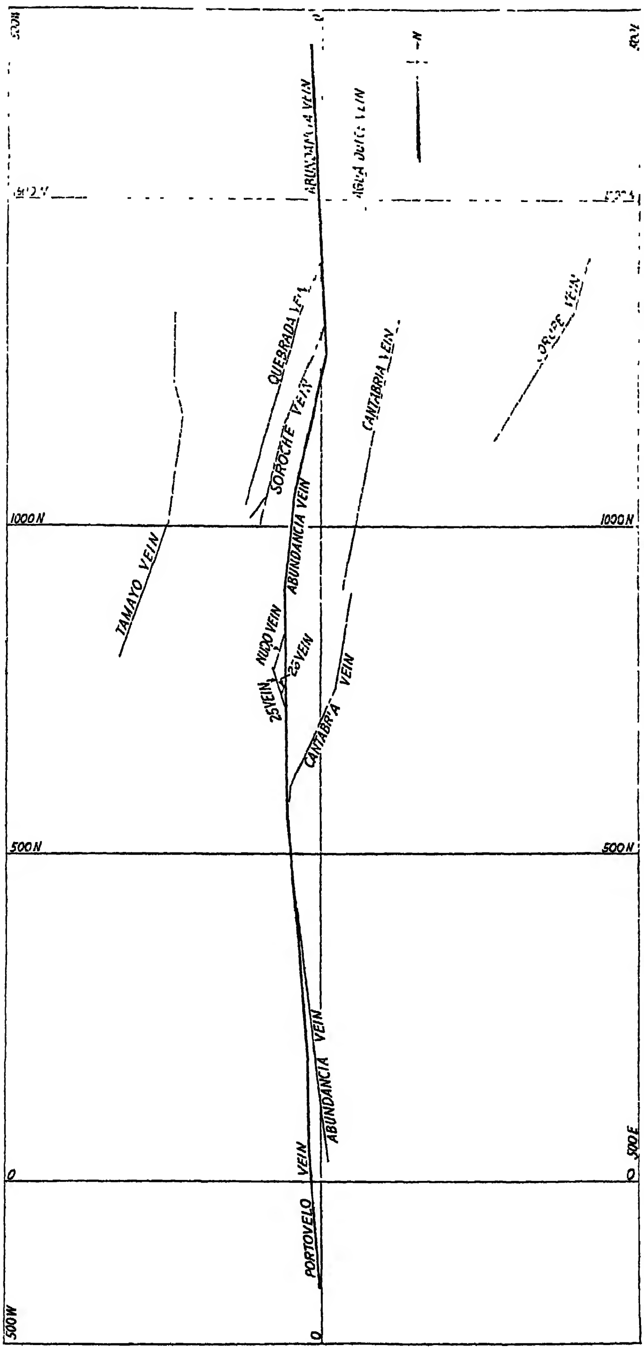

FIG. 1.—VEIN SYSTEM, SHOWING ALL IMPORTANT VEINS AS OPENED ON A LEVEL, OR PROJECTED TO A LEVEL ELEVATION FROM OTHER UNDERGROUND WORKINGS.

ence of vein material. The sheets of quartz and calcite, some minable and others almost barren of precious metals, occur in large lenticular masses. They have the strike of the fault plane, in general N. 3° W., and its dip of 65°–68° to the east. These lenses may extend vertically 200 m., while the greatest length opened up on any one level is about 80 m. Stoping widths of 1 to 6 m. are found. The lenses have a rake to the north with dip of 65°–70°, and may be found anywhere along the fault; seemingly, there is no rule as to the position where one may be expected, although large orebodies have been found near the junctions of Abundancia vein with the Portovelo and Cantabria veins. As a rule, the extent of the orebody is less than that of the quartz lens; or in other words, the lens is not all ore. Some lenses have more than one oreshoot, and others are entirely barren of ore. They apex at different altitudes, not all of them cropping at the surface, and pinch out at variable depths. The valuable portion of the vein may lie on the foot wall at one elevation, and on the hanging wall at another, the oreshoot proper consisting of overlapping lenses within the main lens. The difference between ore and waste is not discernible to the eye, hence the necessity for close sampling. True walls are present, with considerable gouge, especially on the foot wall. A false foot wall, characterized by a strong slip, is often found. Between it and the true foot wall, there is commonly from 0.30 to 0.75 m. of low-grade quartz. The walls will not stand after mining the vein, but break well back on both sides to parallel slips within the fault zone. Many tongues of quartz follow minor fractures out into the hanging wall away from the main body, but never beyond the hanging wall of the fault zone proper.

Twenty-six vein is short, only 100 m. long on the longest level. It has only one oreshoot with a depth of 150 m. In the upper part, it is a hard white quartz averaging 1 m. wide. Calcite appears in depth, where it grows wider and is of low grade.

Cantabria vein is a fracture making away from Abundancia fault to the northeast. It holds many orebodies, generally lenticular in form. The ore extends from 150 to 180 m. below the surface. The most intense mineralization is near the fault; here the company worked to a depth of 200 m., the length of the oreshoot being 85 m. and its width 4.5 m. Cantabria is the most base of all veins worked, galena, sphalerite, chalcopyrite and pyrite being present. Its walls are not well defined except at the junction with Abundancia fault.

Soroche vein makes into the Abundancia fault from the foot wall side; it has made several good sized orebodies. It has a banded structure of alternate ribs of hard and soft quartz with little calcite or sulfides. There are no walls. Stringers of quartz lead out from the main body into the foot and hanging walls and many overlapping kidneys of ore are found. The vein itself is badly broken and has numerous pockets of high-grade ore. The average stoping width has been 2.74 meters.

Tamayo vein has not been connected with Abundancia fault so far. Its ore lies in overlapping, slightly offset lenses of quartz that strongly resemble those of the Soroche vein. Some sulfides are present and it is strongly oxidized in the upper levels.

The Jorupe vein is composed of very hard massive quartz and the base sulfides. Several oreshoots have been developed, on one of which stoping has been started.

Exploration

The ore deposits of fault planes and fissures of the Zaruma district have a dip of 50° from the vertical, usually to the east. The oreshoots vary in length and width and may apex or bottom at any elevation. They rake usually toward the north. There is no secondary enrichment and values bear no relation to topography. On account of thick soil and undergrowth, outcrops of ore are found with difficulty, except upon the sharp ridges and in the deep ravines, where the surface is worn to bed rock.

Preliminary exploration is generally carried on through adits. Depressions resulting from the cavings of old workings, dumps, etc., have often guided exploration. The abandoned workings are seldom found to contain ore; they were quite thoroughly mined to the water level, but they indicate where ore may be found at greater depths.

Prospect tunnels are driven on the vein or in the walls, depending on the character of the formation. Where the vein is wide, frequent crosscuts are made to the wall; and where it pinches, crosscuts are extended into foot and hanging walls in search of overlapping veins. Following the development of ore on one level, additional levels are opened 30 m. above or below the first, depending on the surface contour, and raises are put up at intervals of 30 meters.

Close sampling is the practice wherever a vein is exposed. Channel samples are moiled from the face and back 1 m. from each other, about 15 lb. being cut per meter. The samples may represent the whole width of the vein, or may be split as the character of the vein changes. The usual records and sample maps are preserved.

In the stopes, the faces are frequently sampled under the direction of the mine superintendent for the purpose of directing the daily work. Once every 6 months, the entire stope surface is sampled and mapped just as the drifts are sampled, and tonnage estimates are made, using 2.9 tons per cubic meter. Because of the indefinite walls at many places, the tonnage recovered is usually larger and the values lower than estimated from moil sampling.

Mining

In the early Spanish days, mining was conducted in the customary crude manner. The ore was followed in depth to water level, usually about 100 ft., leaving low-grade ore for pillars. The ore was carried out,

the men using chicken ladders, steps cut in the walls, or built of masonry. The Spaniards left no ore that can now be mined at a profit.

The first work done by the South American Development Co. was through adits, the principal working adit being known as A level. Many large oreshoots were found above this level and worked out with filled stopes on the rilling system. The usual procedure was as follows: After the development of an orebody by drifting, a lateral was driven parallel to it about 6 m. in the foot wall, and crosscuts were broken through from the lateral to the drift at 20-m. intervals. Raises were then driven through to the surface from alternate crosscuts. The orebody was next silled off from wall to wall, as high as the ground would stand without timbering. Rills were then started from the raises, and the ore stoped above the fill which was introduced from mill holes on the surface. The broken ore was shoveled from the toe of each rill and later was handled through chutes built in the crosscuts. Costs were low by this method.

Some time later, the shrinkage system was adopted. This change was hastened by the fact that the stopes were located farther and farther away from the surface, making it increasingly difficult to get the cheap surface fill. Moreover, this fill was not altogether satisfactory. It was composed of surface wash, badly weathered, mostly very fine and not possessed of sufficient body for good fill. As it was worked into the stopes in quantities, a mill hole was formed around the top of each filling raise or chimney. During the rainy seasons, these mill holes had to be bulkheaded off to hold back the accumulated mixture of surface wash and water.

In order to work the deposits on the Abundancia and Portovelo veins lying below A level, the American shaft was sunk; it is collared in the hanging wall a short distance south of the intersection. This location was advantageous because it was close to many large, high-grade stopes on both leads, but disadvantageous because of the temporary loss of good ore left in the shaft pillars. At this time, stoping by shrinkage reached its maximum development. Shrinkage stoping of the narrower oreshoots on the northern end of Cantabria vein was a success. The walls were firm, standing very well, and the ore suffered but very little dilution through the admixture of waste. On the wide orebodies of the Cantabria vein near its intersection with the Abundancia fault, and on the orebodies of the Abundancia and Portovelo veins, shrinkage stoping was not so successful. After drifting and opening the veins to their full width, the stopes were either cut out for a height of about 4.57 m. above the rail, timbered, and breaking carried up on the timber; or box-hole raises were driven for chutes, the raises being lengthened both ways on the strike of the vein, connected a short distance above the level, and the stope carried up from them. This system was efficient as far as low

breaking costs were concerned, but a large percentage of the ore in any orebody was lost. In the first place, it was impossible to delineate clearly the outlines of an orebody as irregular as these; and in the second place, much ore was lost through caving of the walls in great slabs so that the ore was hung up and left behind in the stopes. Even when the walls did not cave, there was a loss in the ore that clung to the foot wall when drawing the stopes. There was also dilution of the ore through admixture of waste from the walls.

In late years the shrinkage system of stoping has been abandoned almost entirely, there being a reversion to filled stopes of the horizontal cut with back-fill type, or the modified rill with waste obtained from development on the upper levels or broken in waste raises at the head of the rill. Costs are higher, because of the extra cost of breaking and handling fill and the extra handling of ore in the stopes, but a close examination of the records shows that up to 40 per cent. more ore is obtained from a given orebody. The ore is also kept much cleaner, as considerable waste is sorted out and left in the stopes for fill. In high-grade stopes, however, there is a chance for loss through the mixture of fines with the fill. After blasting rich ore, it has paid, in some instances, to remove the upper layer of fill, to a depth of 0.61 m. (2 ft.) and send it as ore to the mill. The development of the Soroche mine, or that part of the property lying adjacent to the northern end of A level and extending about 152 m. to the surface above, which has taken place during the last four years, is chiefly responsible for the change in practice. The orebodies in this mine are characterized by the lack of definite walls, the overlapping lenticular structure being decidedly pronounced, and the walls extremely shattered and heavy. Many of the stopes must be closely timbered and then filled tightly as soon as possible to prevent caving. The finding of so much ore in the walls led to the adoption of this method on the other veins with similar results.

Previous to the installation of the modern cyanide plant, which began operations in April, 1919, the milling practice was straight amalgamation, followed by a rough classification and cyanide leaching of the sands. The extraction was low, as well as the tonnage. For some years, selective mining was carried on and the mill heads kept at $18 or better, so that operations might be carried on at a profit. This resulted in taking the cream of an orebody, with much lower grade material left unbroken as pillars and on the ends of the old stopes. A considerable part of the present tonnage is derived from working the edges of these old stopes; the cut-and-fill system is admirably adapted to the extraction of this ore. The operation, on the whole, is similar to that of any cut-and-fill stope, except that the old adjacent stope must be filled as well as the actual working stope; see Fig. 6.

Mining Methods

Development Plans

The combined Portovelo and Soroche mines are now worked through a shaft and two adits. The collar of the American shaft and A level are at the same elevation. The levels above A level, known as B, C, D, etc., are driven 30 m. apart. Mining is being done on D and E levels, while A, B, and C are on development work. Ore from above A level is passed through rock chutes to that level and transferred to the mill by mule train. Waste is used in filling stopes either above or below A level.

Eight levels are turned off from the American shaft at intervals of 30 m. each. The shaft has two compartments down to the seventh level, then three to the ninth, and two below to the depth of 320 m. Sinking will continue to a depth 343 meters.

The greatest lateral development is on the third level, which has been opened up for 420 m. to the south of the American shaft and for 920 m. to the north. The levels of the Soroche mine overlap those of the Portovelo mine (American shaft) on the south and extend 360 m. beyond the north face of the third level. Increased depth is gained by following the Abundancia fault to the north, the country rising in that direction. The combined length of the lateral openings of the Portovelo, Soroche, and Jorupe mines, including the outlying prospects, is over 35 kilometers.

The usual method of development is to drive a crosscut, to the vein to be developed, from the American shaft, in the case of the Portovelo mine, or from the working raise in the Soroche, perpendicular to the Abundancia fault, then turn off drifts both north and south. Drifts are run on the foot wall with crosscuts to the hanging wall at frequent intervals, in high-grade ore; and at greater intervals in low-grade ore or waste. When the faulted area is barren, laterals are often run paralleling it, with occasional crosscuts to the fault for exploration. This avoids the necessity of timbering, which must always be done when following the faulted area.

When an orebody is outlined, a raise is generally driven to the level above, for the purpose of further developing the block, for ventilation, and to serve as a working raise for stoping. The method of stoping to be used is then decided. If the orebody is to be stoped through a lateral, this is driven in the foot or hanging wall, and chute raises are broken through; otherwise the stope is cut out or box-holed over the drift for stoping by shrinkage or by the cut-and-fill method. During the last ten years, about 25 tons of ore have been developed for each meter of development.

Nearly all development is done by contract. The contractor pays for all supplies, including timber, but excepting tools, drill steel,

oil and air. Track and pipes are laid by the company; it also pays for the timbering in headings.

Sinking, Stations, Pockets, Etc.

Native labor is extremely inefficient on sinking, 10 m. per month being about the maximum attainable. For drilling No. 55 Clipper, No. 95 Waugh drills, and B.C.R. Jackhamers are used. Blasting is done with 1 in., 60 per cent. gelatine, using ordinary fuse and caps. Local timber will not withstand the shock of electric blasting and delay-action primers are too complicated for the native workmen.

Ordinary shaft sets are used; 7-in. timbers being placed on 6-ft. centers. Formerly bearers were of squared timbers hitched into the walls. At present, bearers are made of concrete, poured in place, and reenforced with old rails and twisted rods. The hoisting compartment is lagged throughout on all sides with 3-in. plank; the other compartments are lagged only where the ground requires it. The guides are of 4 by 6 in. timbers, with notched lap joints, and are lag-screwed to the end plates and centers. Two small auxiliary hoists have been used for sinking, the dirt being raised in buckets to a pocket at a level above. A No. 6 Cameron sinking pump handles the water during sinking operations.

The stations at the levels are small, no car capacity being required. Ore pockets up to 175 tons capacity and waste pockets of 10 to 15 tons capacity are provided at each level. Grizzlies, made of 30-lb. rails and having 1-ft. square openings, cover each pocket.

Drifting

The drilling machines used in the headings are the No. 21 Turbro with 1¼-in. round hollow steel and the No. 50 Clipper and No. 93 Waugh with ⅞-in. hollow hexagonal steel. The Turbro is too heavy for the average native and is only used in very hard ground. Average advance is about 20 m. per month. There is nothing peculiar in the method of advancing the faces, except that in the hard veins, consisting of quartz and massive calcite, a larger number of holes is necessary; the calcite makes the ground tough and hard to break. Headings are driven 5 by 7 ft. in the clear and the average round pulls 1.1 to 1.2 meters.

In addition to the machine work, over one-half of the total advance is made by hand, chiefly because a great saving on compressed air is made, the compressor capacity not being sufficient to do all of the necessary development by machine. Hand work is also cheaper from all other angles. Single-jackers are paid lower wages than machine-men, explosives are used more efficiently, and there are savings on oil, piping, drill steel, drill repairs, etc.

Raise Practice

Most of the raises driven are stoping raises for blocking out the ore and for ventilation; later these serve as stope manways. Other raises are driven solely for ventilation and for permanent manways or safety exits, or for transfer raises for ore or waste. All raises in waste are driven as two-compartment raises; those in ore may have two or three compartments.

Drilling is done with No. 16V Waugh stoper or No. 71 Waugh stoper; 50 to 35 per cent. gelatine is used for blasting. An advance of 50 ft. per month is considered good work. A Little Tugger hoist is part of the equipment for all raises. Two-compartment raises are carried in the country rock, 11 by 5½ ft. over all. These are timbered as raised and lagged with polls or planks. Manway platforms are placed at intervals of 30 ft. The ends of the timber, as a rule, are not notched, cleats being used instead of notches. Centers are placed every 5 ft., and the rearing, or chute lining, is of 3-in. plank as before. Bulkheads are carried over the manway in all cases.

Stoping

The shrinkage method of stoping was formerly used extensively, but as the walls are unsuited to its use, it has now been abandoned almost

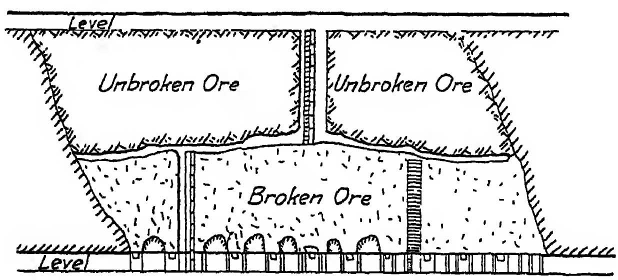

FIG. 2.—SHRINKAGE STOPE, BOX-HOLED WITH A STULLED MANWAY AT ONE END TIMBERED WITH A CRIBBED MANWAY AT OTHER END.

entirely and will not be considered here, except as to the first steps which are practically the same for the cut-and-fill method that is now commonly used. The stopes are either cut out for timber, or box-holed for chute raises and manways; see Fig. 2. With the first method, two cuts are taken out of the back of the drift from wall to wall, leaving the back of the cutting-out stope about 16.4 ft. above the rail. The muck from this first operation is then cleaned up and timbering started. When the veins are narrow, sets of caps and posts are placed, the caps reaching from wall to wall. On the wider veins, timber cannot be obtained long enough for this purpose. The sets are placed and blocked temporarily, then pole lagging is placed back of the posts of all sets, and the space

between the lagging and the walls filled with waste to the level of the caps. Sets are placed 4½ ft. center to center.

Chute mouths of double 3-in. plank are built at every third set. Two manways are generally carried up with the stopes, either timbered in the usual manner or cribbed. When a stope is started by box-holing, the raises are spaced at intervals of 16.4 ft. They are opened 5 ft. long on the level, this length being increased in both directions with each succeeding round, until they are connected about 16.4 ft. above the back of the level, leaving triangular shaped pillars between the chutes.

Practically all of the present stopes are worked by the cut-and-fill method. They are either carried on timber or worked through stoping laterals. In the first case (see Fig. 3) the stope is cut out and timbered in the same manner as a shrinkage stope, except that chutes are built in every fifth set. Cribbed chutes and manways are carried up through

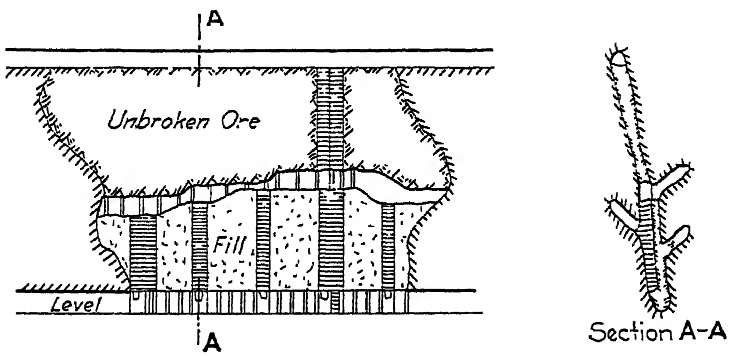

FIG. 3.—CUT-AND-FILL STOPE, WORKED BY BACK-FILLING METHOD, FILL OBTAINED FROM WASTE RAISES IN BOTH WALLS, STOPE PARTLY TIMBERED, CRIBBED CHUTES AND MANWAYS.

the stopes. With soft ores, cribs of round timbers, unlined, are sufficient; with hard ores, the unlined cribs wear rapidly and are a constant source of trouble. Cribs of squared timbers lined with 3-in. plank are better for this class of muck. The round, unlined cribs are carried with the dip of the vein, no offsets being necessary. Cribs of square timber, plank lined, are carried up vertically with offsets, to facilitate the nailing of the lining in a secure manner. When stoping from a lateral, the lateral is driven parallel to the orebody at a distance of 5 m. in the wall; see Fig. 5. The foot wall is the preferable location, but in the case of parallel veins separated by a small width of country rock, one of the laterals must be driven in the hanging wall. The chutes are raised from the laterals to the stopes each 8 m., and are carried up with cribbed timbers. When working stopes in heavy ground, timber as well as filling must be used; see Fig. 3. Stope timbering is done after the

manner of drift sets, with caps from wall to wall. The posts are stood on footboards placed on the fill, and the sets are 8 ft. high; they are spaced from 5 to 8 ft. center to center. Stringers of 6-in. round sticks are laid from cap to cap, about 1 ft. apart, and blocked down from the back. Filling closely follows breaking in these timbered stopes, the fill being kept close to the back. The posts are lost in the fill, but the caps and

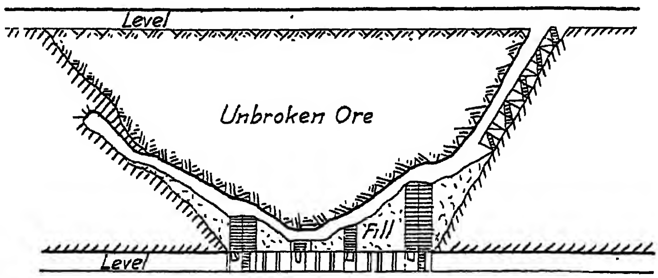

FIG. 4.—CUT-AND-FILL STOPE, RILLED FROM A STOPING RAISE AT ONE END AND FROM A WASTE RAISE IN BARREN VEIN IN OTHER END; STOPE CHUTES AND MANWAYS CRIBBED; ORIGINAL STOPING RAISE STULLED.

stringers are recovered to be used on the next cut. Breaking of ore and waste in the cut-and-fill stopes is done with stoping machines of the No. 16V or No. 71 Waugh type, or with jackhammers of the Waugh No. 95 or Ingersoll-Rand BCR-430 types. As a rule, one or more horizontal banks are carried across the stope followed by filling, which is obtained by driving raises and crosscuts into the foot and hanging walls. A small

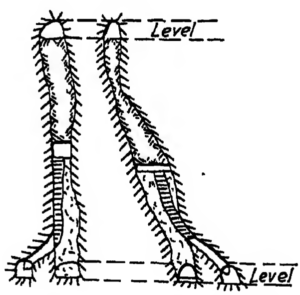

FIG. 5.—STOPES ON PARALLEL VEINS, ONE WORKED THROUGH A FOOT-WALL LATERAL, THE OTHER THROUGH A HANGING-WALL LATERAL.

amount of fill is also obtained by sorting the ore in the stopes. In some of the stopes, part of the fill is obtained from development waste on the level above, this waste being dumped through the stoping raise. When waste is obtained in the latter manner, the stope may be rilled to this raise; see Fig. 4. Other rills are made by breaking waste in the end of the stope by raising on the barren vein, and then rilling to the waste raise;

see Fig. 4. These raises serve the double purpose of providing waste and developing the vein. It is important that a part, at least, of the fill should be broken in crosscuts or raises in the walls, as these workings have the added value of being good prospects. They need not be long, as the overlapping lenses are always found within a few meters; often they are separated by only a shell of waste. The advantages of the cut-and-fill system are 100 per cent. recovery of the ore in any oreshoot; the ore is kept cleaner, as sorting can be carried on in the stope; and there is no admixture of wall rock, as when pulling a shrinkage stope.

The drilling in all stopes is done on a contract basis, the machine men receiving a fixed rate per meter of hole drilled. All holes are spotted by native stope bosses, and measured by them before blasting. Bonuses are paid to those machine men that work steadily and drill more than a

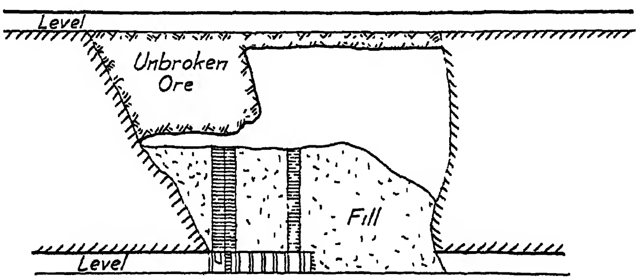

Fig. 6.—Cut-and-fill stope carried up at one end of an old, caved, shrinkage stope.

given meterage per month; the bonus takes the form of a higher rate for all meters drilled. Loading and blasting is done by the drillers under the supervision of the stope bosses.

An analysis of the cost sheets will show that, for the period from 1902 to 1908–9, the direct mining costs, including development, amounted to S/9.40 – S/9.50 per ton milled. During this time, the average yearly development amounted to 925 m. at a cost of S/57 per m. This period represents that in which the large surface stopes were mined by the rilled cut-and-fill method. The ores near the surface were softer, much more easily and cheaply mined, and the surface fill was cheap. Moreover, many of the stopes were above A level, so that pumping and hoisting costs were low.

During the period 1909–19, the direct mine costs were S/8.00–S/8.10 per ton milled; this includes an average yearly development of 1530 m. at an average cost of S/49 per meter. During this time, when shrinkage stoping was at its height, therefore, the mine costs were much lower in spite of increased development. A great part of the tonnage came from the American shaft, so that hoisting and pumping charges were

inevitably higher. The cost of supplies during the latter years of the period, the years of the war, also was high. The bulk of the tonnage, however, came from the enormously large bodies of ore lying within a short distance of the shaft. A few stopes were sufficient to give the desired tonnage, and close supervision was possible; this fact accounts for cheap breaking, tramming, and total mining costs. The great disadvantage of this cheap mining lies in the high loss of broken and unbroken ore left behind in the caves that resulted when the stopes were pulled, or overlooked in the walls during actual mining.

From 1920–22, inclusive, the average direct mining costs were S/12.40–S/12.50 per ton milled. This seems to be a large increase, but can be accounted for in several ways. Shrinkage stoping had been almost entirely discontinued, the cut-and-fill method, by back filling or with the modified rill, having been adopted once more. This method undoubtedly requires more labor, resulting in increased costs. A minimum increase of 20 per cent. in the wages of all mine labor also went into effect at the beginning of this period. The costs of all supplies have continued to be high. Moreover, there has been a big increase in the quantity of development work done. In this period, an average yearly development of 3796 m. at a cost of S/51 per m. was done. While the cost per meter compares favorably with the costs of the preceding periods, the greatly increased meterage shows its effects in the total mine costs. Most of the big orebodies near the shaft had been worked out before this time. The tonnage came from smaller bodies at long tramming distances, or from the edges of worked-out and caved stopes. A great deal of filling had to be put into these old stopes before the remaining ore could be mined. Mining of the orebodies in the heavy ground of the Soroche mine has necessitated the use of a great quantity of timber, the first cost of which, together with its maintenance, constitutes a big percentage of the increase in total costs. More bosses were needed to cover the scattered workings and even then the supervision was not as close. The big advantage of this method of mining lies in the greatly increased tonnage gained from an orebody. All things considered, the cut-and fill method of mining, with the stopes worked through stoping laterals, seems best adapted to the orebodies of the district. A tabulation of costs is as follows:

Period	Method	Cost per Ton Sucres	Approximate Cost per Ton Dollars Gold
1902–09	Cut-and-fill	9.40– 9.50	4.60
1909–19	Shrinkage	8.00– 8.10	3.90
1920–22	Cut-and-fill	12.40–12.50	4.15

Records of Unit Production

The following figures have been obtained from one year of normal operation (1922).

Tons per man per hour for all underground and surface labor exclusive of office	0.0534
Man-hours per ton for all underground and surface labor exclusive of office	18.73
Tons per man per hour for all underground labor	0.0614
Man-hours per ton for all underground labor	16.28
Tons per man per hour for all surface labor	0.4087
Man-hours per ton for all surface labor	2.45

Classification of Labor, Expressed in Percentage of Total

Machine men and helpers, development and stoping	32.00
Mucking and tramming, development and stoping	32.50
Timbering	7.24
Tracking	1.48
Filling (all labor)	8.36
Maintenance	2.71
Sampling	0.72
Hoisting	2.86
Pumping	1.79
Machine shops	0.16
Electricity	0.05
Compressors	0.88
Warehouse	0.94
Tools	0.77
Drill-steel sharpening	4.24
Air-drill repairs	0.38
Piping	1.49
Ventilation	0.15
Watchmen	1.28
	100.00

The percentage labor turnover is unknown, but it is very high.

Records of Unit Supplies

Pounds of dynamite per ton of ore mined	2.24
Pounds of dynamite per meter of development	15.45
Pounds of dynamite per foot of development	4.71

Safety and Welfare Work

No safety organization of any kind is maintained, this work being left in the hands of the mine bosses. Due to the inability of the natives to take care of themselves, close attention must be paid to all working places. Obviously, the accident rate with this class of labor is high; the only surprising feature of it being that serious accidents are not more common. All serious accidents are reported to the local authorities in compliance with a compensation law passed in 1922. The company, however, has

had an agreement with its employees for several years which is much more favorable to them than the compensation law.

The company hospital is the most modern and best equipped in the Republic. An American surgeon and trained nurse, together with an Ecuadorian doctor, are in attendance. All employees and dependent members of their families are given free treatment and medicines for all classes of cases without restriction. Many outside cases are also treated at the company hospital, this work being done for a nominal fee or free of cost. All prospective employees are subjected to a hospital examination, as a result of which they are passed for work or rejected as unfit. Those qualifying for employment are treated for hookworm, and classified as to their fitness for surface or underground employment. Many sanitary measures have been instituted, and strict enforcement of sanitary regulations is maintained.

A large boarding house is operated by the company for the benefit of the Ecuadorian employees, they having the option of boarding with the company or receiving an allowance of 50 centavos per day above wages. This boarding house is operated at a loss, using the amount of 50 centavos per day per man as a basis to figure from. Dwelling quarters at a very low rent are also provided for all employees who wish them.

The members of the staff are quartered in frame and concrete houses; married men being furnished with houses, and bachelors with single rooms. A boarding house is also operated for the benefit of the single men on the staff. House rent, lights, fuel, water, etc., are furnished free, as well as ice and distilled drinking water. Recreation is provided for in the shape of a clubhouse, tennis courts, baseball field, and swimming tanks. There is also a company stable, where staff employees' horses and mules are cared for at a very small charge.

DISCUSSION

Rudolph Emmel, Quayaquil, Ecuador.—All the labor is very inefficient and the bonus system is the only way we can get anything at all. Drilling is done by paying so much per meter of holes. Drifting is done by contract on the basis of so much per meter.

George A. Packard, Boston, Mass.—That accounts for the apparently very low efficiency in man-hours per ton. At Cornucopia and at Jarbidge, the drillers in stopes practically average 2 tons per man per hour, or ½ hour per man per ton; whereas at Zaruma the average is 0.05 ton per man-hour or practically 19 hours labor. Later the author shows that machine men and helpers on developing and stoping make about 32 per cent. of the labor, which would mean 60 man-hours per ton of ore or twelve times the requirement at Cornucopia and Jarbidge. This figure includes both the tons per man per hour in the shrinkage stoping and in the other; what is the production on shrinkage stoping alone?

Rudolph Emmel.—I have no figures on shrinkage stoping here; the figures given include all of the developments and we are doing an abnormal amount of development work. For a mine producing 225 tons, 4 km. a year is a large amount and tends to bring down that figure. However, the labor is very inefficient; it is incredible the small amount of work that one of those men can do in a day.

Fred Hellmann,* New York, N. Y.—My experience in the mines of South America has been entirely different from that. The Chileans would compare favorably with drill men in any part of the world. I consider the Chilean one of the best drill men in the world.

R. M. Raymond,† New York, N. Y.—It is largely a matter of the make-up of the men. Mexicans are rather fond of mechanical work. They make excellent men for running drills, do not mind dust, and work fairly steadily. They are good at any kind of machinery, even running hoisting machines. They are showing themselves adapted to such work in a surprising manner. How do the Bolivians compare with the Chileans?

Fred Hellmann.—The Bolivians cannot compare with the Chileans. They are not nearly as intelligent. The Inca race, as you know, was conquered by the Spaniards, and their present conditions testify to the inaptitude of the race and its weakness.

R. M. Raymond.—How do the Chileans compare with the Kaffirs as to man power?

Fred Hellmann.—The Kaffir must be taught everything when he enters the mine, while the Chilean is much more intelligent. The Chileans are essentially a mining people. You can induce the Kaffir to do things, and under certain conditions he is a wonderful worker. The Kaffir can be taught to run a machine but it will take him much longer than the Chilean. I do not know how you could state it in percentage, but you would get possibly twice as much work out of the Chilean miner as out of the Kaffir.

In Bolivia, there are large mines but the results obtained could never be had by our methods of mining. If an American should go there with the intention of lowering the cost of getting out the ore by using more modern methods, he would soon find his mistake. They mine about as cheaply as is possible, though their methods are somewhat primitive. Access to the mine is usually given by a spiral stairway—the so-called Boca Mina—up which the ore is carried on the backs of the workers. It is hard to believe that you cannot improve on that, but the supply of

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† Professor of Mining Engineering, Columbia University.

labor is so great and the comparative cost of it so small that when you figure the cost of installing and operating machinery in a country that is not mechanically inclined, you find that the present is the better method. Of course mining 50,000 tons of ore a day is another kind of mining; that is done with steam shovels.

Arthur Notman, New York, N. Y.—The report of the Union Miniere du Haut Katanga for 1923 indicates roughly the number of men employed by the company for that year. There was no classification of labor or segregation as to operation and construction given in the report, but taking the whole number of employees as given, an output of 35 lb. of copper per man per day is indicated. Somewhat similar figures for the Chile Copper Co. indicate, for 1923, an output of 115 lb. per man per day. The output of the porphyry copper mines in this country runs from 150 to 190 lb. per man per day. Even after making due allowance for perhaps an extraordinary proportion of the labor on construction work in the Congo, it does seem as though the much higher grade ore and lower wages were pretty well offset by the low labor efficiency.

Fred Hellmann.—You have an entirely different type of labor in the Congo. The northern tribes and the tribes of central Africa are not nearly as husky and strong physically as the natives of the more southern parts. They are much weaker and much more subject to disease and death when they do work, especially under changing climatic conditions.

C. F. Jackson, Skouriotissa, Cyprus.—Labor is our chief problem; the technical problems would not be so bad if our labor did not keep undoing our work. For example, the other day coming out the main intake airway of the mine, we found a canvas had been stretched across the portal by some workmen employed nearby who wanted to shut off the chill air; sometime ago we sealed off a small fire but not long after some native opened a 9-in. hole in the wall with a pick. These people tear out ventilating doors, remove brattices, close doors that are supposed to be kept open and leave open doors that should be kept closed, and pick into pillars that are necessary to the maintenance of important airways and roadways. With good labor the work would be comparatively easy, but the Cypriot is the worst workman in the world to deal with; there is no punishment that he cares anything about so that it is next to impossible to discipline him.

Rudolph Emmel (Author's reply to discussion).—The figures of 2 tons per man per hour at Jarbidge compared with 0.05 ton per man per hour at Zaruma is a comparison of the output of men directly engaged in stope drilling at Jarbidge with the output of all men connected with the work of the mine, underground and surface, at Zaruma. The figures used for comparison should be 0.4 ton per man per

hour at Jarbidge (mining by shrinkage) against 0.05 ton per man per hour at Zaruma (mining by cut-and-fill methods). The mines at Zaruma also are undoubtedly carrying a much heavier burden in the nature of development work necessary than those at Jarbidge.

As to the efficiency of the Chilean miner, one must bear in mind that the Chilean is working under entirely different conditions from those that prevail in Ecuador. The Chilean is the product of the temperate zone, is physically and intellectually much superior to the Ecuadorean miner, and has had the benefit of a much closer contact with American and European labor and methods. We all know that the Mexican, when in competition with American labor, has developed to the point where he works side by side with miners of all nationalities in the United States, and has supplanted the higher priced labor to a great extent throughout the West. Chile is a very cosmopolitan country, is one of the most advanced of the Latin-American nations, and the Chilean workman has undoubtedly benefited therefrom. The Ecuadorean workman physically is a poor specimen, he has generations of hookworm and malaria and other diseases back of him, is working in hot, poorly ventilated mines in a tropical country, and has come into no competition or association with the outside world.

Methods of Mining and Ore Estimation at Lucky Tiger Mine

By R. T. Mishler,* El Tigre, Son., Mexico, and L. R. Budrow,† Douglas, Ariz.

(New York Meeting, February, 1925)

The Lucky-Tiger mine is a silver-gold property, situated at El Tigre, in the northeastern part of Sonora, Mexico, at an elevation of 6000 ft. in the Sierra Madre Mountains. It is 30 miles by wagon road from the Esqueda station on the Nacozari railroad.

Since its discovery in 1903, the mine has produced forty million ounces of silver. The ore has averaged 40 oz. silver and 0.25 oz. gold per ton. Copper, lead and zinc have been of minor importance, having averaged 0.4 per cent., 1.1 per cent., and 1.5 per cent., respectively. The richness of the ore, combined with its occurrence in narrow veins, required the use of the more expensive methods of mining.

Geology

Wall Rock

The wall rock is rhyolite and rhyolite tuff, both of Tertiary age. In rare instances, post-mineral andesite dikes have been intruded along the veins. In the upper part of the mine, the walls were fairly solid. On the lower levels, the walls are usually either kaolinized and soft or silicified and fractured, both conditions requiring closely filled stopes or much timber.

Veins

The veins have been deposited along fracture planes in the volcanics. There are three principal veins, about 600 ft. apart, all striking nearly north-south and dipping steeply to the west. Table 1 records the approximate dimensions and assay of the veins. Although the average

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† Vice-president and General Manager, Lucky Tiger-Combination Gold Min. Co.

depth below the surface is recorded as 1100 ft., it is probable that the lowest workings are 3000 ft. below the former top of the volcanic series.

TABLE 1.—*Approximate Dimensions and Assay of Veins*

Vein	Developed, Length, Feet	Developed, Average Depth, Feet	Ratio Area Stoped to Area Developed, Per Cent.	Average Width of Ore, Feet	Average Assay Ounces, Silver	Stoping Width, Feet	Silver Assay to Stoping Width
Tigre	6,000	1,200	35	1.9	68	3.5	40
Seitz-Kelley	5,000	1,000	11	1.1	105	3.0	44
Sooy	1,500	1,000	20	1.4	80	3.5	37
Totals	12,500						
Averages		1,100	25	1.7	73	3.4	40

Note: All of the vein assaying 15 oz. or more is considered as ore.

Ore Deposits

As shown in Table 1, a quarter of the area developed has been sufficiently rich to warrant stoping. The individual orebodies, within the veins, are irregularly lenticular, usually with the horizontal axis longer than the vertical. The lenses range from 500 by 2000 ft. to 10 by 50 ft. Usually the veins are continuous between deposits, the vein filling being too low grade to mine. Branch and parallel veins are common, especially on the lower levels. The width of the deposits rarely reaches 20 ft. and in such cases the veins usually consist of multiple stringers separated by waste. Usually, the deposits are less than stoping width, requiring stripping or breaking of waste and ore together. The average width of all deposits has been 1.7 ft.; the average stoping width, 3.4 feet.

Ore

The ore consists of intergrowths of sphalerite, galena, pyrite, chalcopyrite, tetrahedrite and stromeyerite, in a gangue of kaolinized or silicified rhyolite. The silver lies mostly in the tetrahedrite and stromeyerite, though all the sulfides contain commercial quantities of silver. The average grade of the clean sulfides is 550 oz. silver per ton; that of the ore seams (1.7 ft.) 73 oz. per ton; and that of the ore delivered to the concentrator, 40 oz. silver per ton. Thus the dilution is shown to be 82 per cent. This is partly low-grade ore that can be treated at a profit, but most of it is practically waste, broken to give stoping width. Stripping is practiced wherever possible, but when both vein and walls are friable, loss in the fill is avoided by breaking ore and waste together and sending both to the concentrator. In mining rich ore, low dilution must be subordinated to maximum recovery.

Development

The upper part of the mine has been developed through adits. The lowest, No. 7 adit, was driven one-half mile to cut the vein and another one-half mile along the vein to the principal deposit. Below No. 7 level, the mine is developed through underground shafts.

Levels and raises are usually driven at 100-ft. intervals; drifts are commonly 7 by 5 ft. Raises are 8 by 4 ft., and are provided with chute and ladderway, with partition between. A few crosscuts are driven during preliminary development, but most crosscutting is postponed till stoping commences, in order to provide fill for the stopes. Machine

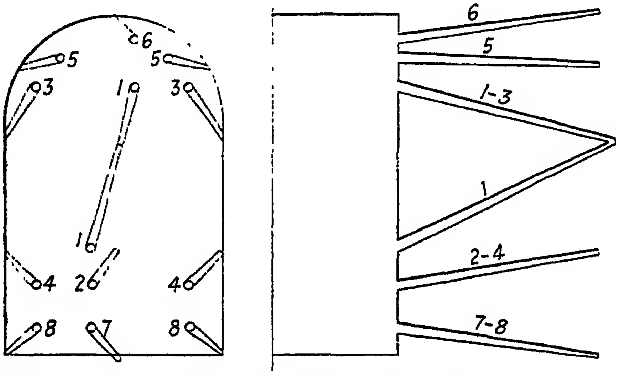

Fig. 1.—Thirteen-hole round, numbers indicate rotation of firing; in hard ground, holes are drilled upward to meet those indicated as 3.

drills are usually employed in development. The round most commonly used is illustrated in Fig. 1.

Practically all development work is done by contract. The prices range from \$4 to \$9 (United States currency) per foot, depending on the hardness of the rock. The contractor pays for his explosives and trams his broken rock to the nearest station. When timber is required, he is sometimes paid \$7.50 extra per set for enlarging the drift and erecting the set. The earnings of the contractors and their men average \$3 per day.

Diamond-drill holes are driven into the walls at intervals of 500 ft. and to depths ranging from 400 to 1000 ft. One excellent deposit has been found by this means.

Stoping

Stoping methods depend on the character of the walls and the width of the veins. Shrinkage stoping is used where the walls are firm and the vein is over 2 ft. wide. Chutes are erected at 25-ft. intervals, and the bottom of the stope is securely timbered and lagged; or, if the bottom

of the block is poor, pillars are left between chutes, up to the limits of the ore. The back of the stope is carried horizontally. Stoping machines are usually employed for drilling; though, when the vein is wide, drifting machines may be used to secure the advantage of horizontal holes. About one-third of the ore is continuously drawn, to maintain the surface of the fill at a convenient level for drilling. Every effort is made to break all boulders and to support all slabs that might fall while the ore is being drawn, but in spite of all precautions, boulders, slabs, and even timber, become embedded in the fill and eventually block the chutes. Often dynamite fails to dislodge such obstructions and it is necessary to cut the lagging above the drift in order to finish drawing the stope. Rather imperfect records, kept over a period of two years, indicate that the cost per ton for shrinkage stoping is practically the same as for other methods.

Open stopes are employed where the ore extends only 20 or 30 ft. above the level, and the walls are firm. The miners drill on platforms supported by stulls. The ore falls upon the track, where boards are laid between and alongside the rails to aid in shoveling.

Underhand stoping is used where the ore extends only 10 or 20 ft. below the level. Usually the broken ore is shoveled or wheeled to development raises; in rare instances it is hoisted to the level above.

Cut and Fill Stopes

Cut and fill methods are most commonly used. They are especially suitable where the veins are narrow or the walls soft. When the veins are 2 ft. wide or less, the stopes are started in the same way as shrinkage stopes, except that chutes are established at 50-ft. instead of 25-ft. intervals. The backs of the stopes are carried horizontally. The ore and waste are blasted separately, cowhides being laid to receive the former. The broken ore is carefully sorted and all fines are shoveled into the chute. The coarse waste is left in the stope. When all the ore is removed, the skins are taken up and sufficient waste is blasted from the walls to keep the stope filled. The broken waste is sorted to remove pieces of ore, but fine ore unavoidably enters the fill. In very rich veins, to minimize this loss, ore and waste are blasted together upon hides, and all the fines are shoveled into the chutes, only the coarse waste being left in the stopes.

In narrow stopes, chutes are formed of two lines of stulls, spaced 5-ft. centers and lagged on the outside to support the fill. In wide stopes, cribbed chutes, at 25-ft. intervals, built of 6-in. round timber, are used.

When the vein is over 3 ft. wide, the stopes usually become too wide and dangerous when all the fill is blasted from the walls; therefore diagonal raises are driven into the walls, or waste from development work is

dumped into the stope through the development raises. In the latter case, the chutes at the ends of the stope cannot be employed as ore passages, and wheelbarrows must be used for handling both ore and waste. Under such circumstances, flat-backed stopes are used only when the ore requires much sorting, or the waste can be blasted separately and left in the stope.

Inclined cut and fill stopes are employed when the vein is wide and requires no sorting. A single chute is established in the center of each 100-ft. block, from which the stopes extend diagonally upward, at an angle of 40°, to the development raises at each end of the block. The ore falls upon inclined plank floors, nailed to stulls, and slides of itself to the central chute. Temporary timber grizzlies, over the chutes, retain large boulders until they are broken. When vein and walls are fairly firm, slices as thick as 10 to 12 ft. are commonly taken. After all broken ore is withdrawn, the floors are removed and waste is dumped into the raises from the level above. This runs into the stope without shoveling and establishes an angle of repose parallel to the back. When it reaches a distance of 4 ft. from the back, the floor is again laid and blasting of ore is resumed. This method is cheap and occasions little loss of ore in the fill, but practically no sorting is possible in the stopes.

Contract Stoping

Where the ore occurs in one or more narrow veinlets and close sorting is necessary, contracts are often made with the miners to break and deliver the ore. Payments are based on the number of cars and the silver assay. Four rate scales are in effect, the scale selected for each stope being determined by the anticipated width and hardness of the vein. The rates range from $1 (U. S. currency) per car for 30-oz. ore, in wide veins, to $7 per car for 100-oz. ore, in narrow veins. The car holds ¾ ton.

The contractors exercise great care in breaking and sorting, in order to secure maximum rates, therefore the grade of ore from contract stopes is usually double that from similar stopes operated by the company. Contract stoping often makes it possible to mine ore that otherwise cannot be worked at a profit.

The contractors invariably use the flat-back cut and fill method, leaving the sorted waste in the stope. They are guaranteed $1 per day per man; when barren zones occur in the backs of their stopes, they are often paid for raises on a footage basis, until good ore is again encountered. When fortunate, they often earn as much as $5 (U. S. currency) per day per man. The average earnings are about $2.50 per day. The system is used throughout Mexico and gives good results with Mexican miners.

Synopsis of Stoping Methods

Shrinkage.—Where walls are firm and vein over 2 ft. wide.

Flat-back Cut and Fill.—Where vein is less than 2 ft. wide (often worked by contract); where walls are soft and vein requires sorting.

Inclined-back Cut and Fill.—Where walls are fairly firm, and vein is wide and requires no sorting.

Open and Underhand Stopes.—Where ore extends only short distances above or below level.

Drilling

All drilling in company stopes is done on contract. Hand drillers receive 12½ or 17½ cents (U. S. currency) per foot and machine drillers receive 5, 7½, 10, or 12½ cents per foot, all rates depending on the hardness of the rock. Hand drills are used on fairly soft rock, where the ore must be broken separately from the waste; machine drills are used when the ore is hard and there is little danger of the waste's falling with the ore. In soft and medium hard ore, hand drilling is cheapest; in very hard rock, the advantage lies with the machine drills. Table 2 gives a comparison of the cost of hand and machine drilling in medium hard rock.

Table 2.—*Hand Drilling vs. Machine Drilling in Medium Hard Rock*

	Cost per Foot, U. S. Currency	
	Hand	Machine
Drilling, labor (contract price)	$0.175	$0.107
Power		0.110
Sharpening	0.030	0.058
Nipping	0.025	0.046
Pipe lines		0.049
Drill repairs and renewals		0.065
Totals	$0.230	$0.435

Though the contract price of machine drilling is only 60 per cent. that of hand drilling, the total cost, including power, pipe lines, repairs, etc. is almost double. The greater depth of machine holes, which increases the breaking power per foot, introduces a factor in favor of machine drills, which might easily offset their higher cost per foot drilled. Contract prices in development work furnish a fairly exact means for comparing

the relative cost of hand and machine work. In soft rock, the contract price is the same for hand as for machine drilling; in medium hard rock, it is 10 to 20 per cent. lower for machine work, but this advantage is more than offset by extra mechanical charges of machine drilling. In soft and medium hard rock the only advantage of machine drilling is increased speed in advancing the face; in very hard rock (hardness 5 or over) the advantage of cost lies with the machine drills. These comparisons apply only with labor at $2 to $3 (U. S. currency) per day.

About 25 machines are in use, practically all of Ingersoll-Rand manufacture. In drifting, 148 and 448 Leyners are used; in stoping, CC11 and CCW11 stopers; and in sinking and plugging, DCRW13 and BCRW430 Jackhamers. Almost all are wet machines. In very hard rock, wet stopers drill twice as fast as dry stopers and the steel runs twice as far.

Compressed air is supplied under 85-lb. pressure by one Twin Angle Compound WN4 Sullivan compressor and two Tandem Compound WH2 Sullivan compressors. The capacity of the three combined is 2000 cu. ft. of free air per minute. The air is piped to the mine through a 6-in. main; 2-in. laterals are used on the various levels. Water, under 50-lb. pressure, is piped to most of the drills. A few pressure tanks are used for isolated drills.

A standard round for drift work, employing a wedge cut, is illustrated in Fig. 1. Rounds employing the same type of cut are used in shafts and raises.

Drill Sharpening.—The machine drills are sharpened in one Sullivan and two Ingersoll-Rand sharpeners. The drills are heated in a Denver Fire Clay furnace. After forging, the bits are annealed in quicklime, reheated in a second furnace, and quenched in water. All work is done on the day shift, the daily average of drills sharpened being 500. The sharpening force consists of three sharpener operators, two drill heaters and one temperer. The cost of sharpening averages 7 cents per drill.

Timbering

Native timber is used exclusively. Delivered at the mine, round pine timber costs $20 (U. S. currency) per thousand, pine plant $45 per M, and oak lagging (4 in. by 6 ft.) 25 cents each. The pine is soft, about two-thirds as strong as Oregon pine.

The type of drift set used is shown in Fig. 2. The diagonal framing between cap and post develops the full strength of the cap. The timber is framed by hand at a cost of $1 per set. In damp drifts, water soaks up the posts for about 18 in. and causes excessive rotting just above the wet portion, therefore 3 ft. at the bottom of all posts is dipped in creosote or carbolinium. All timber for the shaft and permanent haulageways is completely creosoted.

In stopes, both stulls and square-sets are used, depending on the width of the stope. The square-sets are of standard design with the caps butting upon each other.

TRAMMING

Most of the tramming is done by contract. On all levels except the main haulage, the ore is trammed in 16-cu. ft. rotary dump cars. In tramming from chutes the prices are 5 and 7½ cents per car, depending on whether the distance is less or more than 500 ft. When the cars are loaded by shoveling, the price is 12½ cents per car. The cars are counted by toolroom attendants at each station. They record the cars trammed by each man and the cars trammed from each stope or other

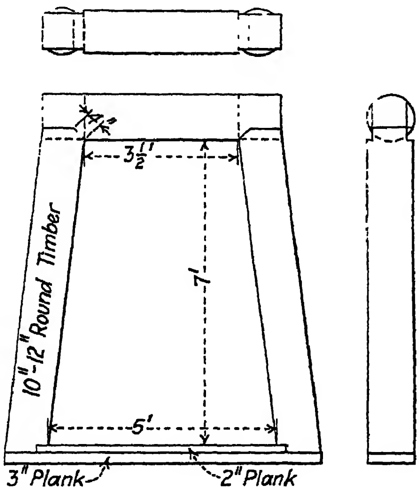

FIG. 2.—TUNNEL SET.

working face. The latter record is consolidated for production statistics. The count of the toolroom attendants is checked by counting the skips hoisted from each level.

On the main haulage level, the ore is trammed in 32-cu. ft. gable-bottom cars and in 16-cu. ft. rotary dump cars. Trains of 10 to 30 cars are hauled by a 3-ton trolley locomotive. The distance from shaft pockets to mill bin is 6000 ft. The contract price for loading, tramming and dumping is 5 cents (U. S. currency) per ton, settlement being based on the automatic scale reading at the concentrator. The total cost of tramming, including power and repairs, is 13 cents per ton, or 12 cents per ton mile.

Hoisting

Two-thirds of the present production is hoisted through an underground shaft at the south end of the Tigre vein. The shaft is timbered with standard shaft sets, spaced 5-ft. centers, and is divided into two 4½ by 5 ft. hoisting compartments and one 3 by 5 ft. pipe and ladderway. Between levels 7 and 10, the shaft is inclined, at an angle of 54° from the horizontal. Below No. 10 level, the inclination changes to vertical. The change in inclination is made on an arc with a radius of 50 ft. In the inclined section, the skips ride on 20-lb. rails. In the curved and vertical sections, the wheels run between timber guides, reenforced with steel wearing strips. The guides were first made of 6 by 6 in. timber, but those proved too light on the curve and 6 by 12 in. guides were substituted, and have been satisfactory. Iron idlers, equipped with roller bearings, support the cable around the curve and in the inclined section of the shaft.

Ore and waste pockets, from 50 to 150 tons capacity each, are provided below each of the lower levels and above the haulage level. A hinged gate makes possible the dumping of either skip into either of the upper pockets.

The skips are 32 cu. ft. capacity. The cables are special steel, ¾ in. in diameter. The hoist is geared to a 50-hp. slip-ring induction motor. Power is transmitted underground through submarine cable, at 2200 volts, and is transformed at the hoist to 440 volts.

The hoist is operated on three shifts and handles 200 tons of ore and waste per day, as well as all timber, steel and supplies for the lower levels. Hoisting costs 16 cents (U. S. currency) per ton of ore and waste hoisted.

Sampling and Estimating

Development Work

In sampling development work, dilution is accepted as an unavoidable evil. Samples are cut over full stoping width (3½ ft. on the Tigre and Sooy veins, and 3 ft. on the Seitz-Kelley vein); if the vein is over stoping width, the full width is sampled. Each channel across the back is divided into as many samples as there are varieties of ore and waste, and the exact width of each is noted. Usually only two samples are taken, one of ore and another of waste. Widths and assays of the various seams are recorded in the engineering office and the average assay to stoping width is calculated. Where the ore is rich and easily recognized underground, it is believed that this method gives a more dependable average assay than is possible when a single sample is cut across both ore and waste.

All drifts, raises and winzes are sampled at 5-ft. intervals. The samples are cut from the backs of the drifts and from alternate sides of

the raises and winzes. The backs of stopes are sampled in the same manner whenever the chute samples show that the ore produced is lower than milling grade.

Assay Map

The assay map is a combination plan and section. The various levels are shown in plan, with sufficient space between them to permit drawing longitudinal sections of the intervening ground. The sections represent as nearly as possible the plane of the vein, so that the dimensions of the various blocks may be readily scaled from them. The scale of 40 ft. per inch has been found most satisfactory. The assays, figured to stoping width, are plotted on the map. Gold and silver are shown to left and right of level or raise, and the width (if more than stoping width) between.

Calculation of Ore Reserves

Ore reserves are estimated as of January 1 and July 1 of each year. The backs of all stopes are surveyed on those dates and plotted on the assay maps. If the back has not been sampled, the average assays along the levels above and below are weighted inversely to their distance from the back, and the average thus obtained is taken as the assay of the back. The same general principle is followed if the assays on any side of the block indicate that side should be left as a pillar. If all the assays surrounding a block represent stoping width, their arithmetical average is taken as the assay of the block. The tonnage is calculated by multiplying the area of the block by the stoping width and dividing by 11.5 (the cubic feet per ton factor). When any of the assays around the block represent more than stoping width, the excess width must be considered in figuring the average assay and tonnage.

If blocks are developed on less than four sides, it is customary to consider that the ore extends 30 ft. from the drifts and raises. The same assumption is made concerning ore below the lowest level. However, the depth of such blocks is never allowed to exceed one-third of the length. Estimates of partly developed blocks are regarded as provisional and are recalculated when the block is completely developed.

The full assay of each sample is employed. The modification of high assays is warranted when the determination of ore reserves is based on only a few samples, but when several thousand samples are taken, abnormally high assays will be offset by those abnormally low and the average will be more nearly correct if the high assays are not modified. When the valuable metals occur in minerals of higher specific gravity than those of the gangue, the average assay as calculated above will be lower than that of a single channel sample accurately cut across full stoping width.

Usually this difference is of theoretical rather than practical significance and can be regarded as introducing a slight safety factor into the calculation of average assay.

To illustrate the accuracy of the above method of sampling, averages have been prepared of two series of alternate assays along four of the principal levels on the Tigre vein. Each series represents sampling the levels at 10-ft. intervals; see Table 3. In all, 1316 assays were tabulated, ranging from 0.2 oz. to 440.0 oz. silver per ton. The average assay was 36.6 oz. Assuming this average correct, the average sampling error of each of the alternate series was 5.3 per cent.

TABLE 3.—*Averages of Alternate Assays along the Tigre Vein as Developed to Date*

Level	Number of Assays	Average of Assays, Ounces Silver per Ton			Error, Per Cent.
		Odd Numbered Assays	Even Numbered Assays	Odd and Even Numbered	
7	428	35.4	41.4	38.4	7.8
8	368	34.3	31.9	33.1	3.6
9	288	36.0	42.3	39.2	8.0
10	232	32.0	39.7	35.9	10.6
Total..........	1316	34.7	38.6	36.6	5.3

Maximum assay, 440 oz.; minimum assay, 0.2 oz.

Another indication of the accuracy of sampling is seen in a comparison of assays of reserves with assays of production. During the past 14 years, the estimated ore reserves have averaged 34.0 oz. silver per ton; the ore mined, during the same period, has averaged 37.1 oz. per ton. The ore reserves having remained fairly constant, the discrepancy of 3.1 oz. in favor of the ore mined indicates that the assays of the reserves were underestimated 8.4 per cent.

A precise comparison of estimates and production is possible only by recording, over a number of years, the original estimates and the production of each block of ore. Such records were started at Tigre a few years ago. They do not yet cover a sufficiently long period to yield valuable comparisons, but they do indicate an underestimation of both tonnage and assay.

Sampling Shrinkage Stopes

In shrinkage stopes, weekly samples are taken across the fill at 10-ft. intervals. The width and distance from the end of the stope are recorded for each sample. The samples, corresponding to the same distances,

are combined for the entire month and assayed. The back and surface of fill are surveyed at the end of each month and plotted on stope maps. The stope maps, together with the monthly widths and assays, are employed in calculating the reserves of broken ore.

Chute Samples

Grab samples, consisting of two double handfuls, are taken from each car of ore as it is loaded. Composites from important chutes are assayed daily. From chutes producing small tonnages, and from development faces, the samples are combined for periods of two or three days. The assays of chute samples are closely watched in order to maintain the grade of ore sent to the concentrator.

Tonnage and Assay Report

The assays of the grab samples, and the record of cars trammed, furnish the basis for the monthly tonnage and assay report (Fig. 3). The cars and cars plus assay are summed for each chute. The chutes are combined in accordance with the stopes they serve, ore from development is included, and the total cars and cars plus assay are summed for the whole mine. The actual tonnage delivered to the concentrator is divided by the total number of cars, to secure a factor by which to calculate the actual tonnage from each stope and face. The gold and silver produced from each stope, and face, are estimated similarly, in ounces. The gross value is calculated from current quotations. The corrected tonnage, metal content, and gross value of the ore from each stope are tabulated by months and by years; thus, when a stope is exhausted, a fairly accurate record of its production is available.

THE TIGRE MINING COMPANY, S. A.
ENGINEERING DEPARTMENT
MINE TONNAGE AND ASSAY REPORT

MONTH OF November 192 4 PRICE OF METALS, AU 20.67 AG 0.69.

Level	Stope	Factors			Ore Broken					
		0.762	0.815	0.691	Tons	Assay		Contents		$ U. S. Currency Gross Value
		Cars	Cars × Assay Au	Cars × Assay Ag		Au	Ag	Au	Ag	
9	106–117	550	115.50	21,450	419	0.22	50.4	94.13	14,822	12,173
9	117–130	224	33.60	8,960	171	0.16	51.2	27.38	6,191	4,838

FIG. 3.—MINE TONNAGE AND ASSAY REPORT.

The factors used for correcting the tonnage and metal content of the ore derived from the various stopes furnish an interesting side light on the

U. S. CURRENCY

TONS AT 2000 POUNDS

THE TIGRE MINING COMPANY

FINANCIAL STATEMENT OF STOPES

Au 20.67

Ag 0.69

MONTH OF Nov. 1921

Level	Stope	Mill Ore		Mining Costs										Milling and General	Tailing Loss		Marketing	Total Deduction Mill Ore	Total Profit or Loss	Total Profit or Loss per Ton
		Tons	Gross Value	Miners	Air Drills	Miners and Air Drills	Mech. Supervision Sampling	Timbering	Hoisting and Pumping	Sorting	Gen. Tramming	Electric Haulage	Total Mine Costs		Assay Per Cent.	Total				
9	106–117	550	12,173	110	330	440	435	495	127	154	204	60	1,915	2,030	5.5	669	2,170	6,784	5,389	9.80
9	117–130	224	4,838	67	159	226	177	224	52	63	83	25	850	826	2	258	864	2,798	2,040	9.10

FIG. 4.—FINANCIAL STATEMENT OF STOPES.

accuracy of counting cars and of grab sampling. The tonnage factor seldom varies more than a few per cent. on either side of 0.75. Since the cars actually hold ¾ ton, the tonnage records can be regarded as very accurate. The silver factor is usually a little low. For the past three years it has averaged 0.68, when it should have averaged 0.75. This represents a sampling error of 9 per cent., which is not considered unsatisfactory for grab samples. The discrepancy is probably due to the tendency of the rich ore to break fine and to the natural inclination of the trammers to take the fine ore rather than the coarse in their grab samples. Perhaps, also, the subconscious desire to secure a good sample may contribute to the discrepancy.

Financial Statements of Stopes

The profit or loss from each stope is estimated monthly in the financial statement of stopes (Fig. 4). The tonnage and gross value of the ore from the various stopes and faces are taken from the monthly tonnage and assay report. The tailing loss corresponding to the ore from each section of the mine is determined by periodical laboratory tests. The numbers of shifts worked in each stope, by hand drillers, machine drillers, timbermen and shovelers, are taken from shift boss reports and consolidated for the whole month. The total costs of drilling, shoveling and timbering, derived from the monthly mining cost statement (Fig. 5), are distributed to the various stopes on the basis of shifts worked. The cost of maintaining nonproducing stopes is distributed to the producing stopes on a tonnage basis. Costs in contract stopes are taken directly from the contract liquidations. Mechanical charges are distributed in proportion to the total cost of machine drilling. Supervision, filling, hoisting, pumping, shoveling, tramming, milling and general are all distributed on a tonnage basis. Marketing charges are distributed according to the gross value of the ore. The sum of all deductions, subtracted from the corresponding gross value of the ore, gives the profit or loss for the stope.

The stopes are grouped according to natural divisions of the mine—Upper Tigre, Lower Tigre, Sooy, Upper Seitz, Lower Seitz, Santa Maria, etc. Development costs are distributed to these same divisions. The total profit or loss from each division is then calculated. Thus at the end of each month a record is available, showing the profit or loss from each stope (with development cost disregarded) and from each division of the mine (with the development cost included). It is believed that this method shows, much better than mere assays, when a stope or division requires special attention, and when they should be stopped.

No pretense of extreme accuracy is made for this report. It is based on many approximations. It must be drafted by an experienced engineer

THE TIGRE MINING COMPANY, S. A. — DEVELOPMENT AND MINING COSTS

ORE PRODUCED 6079.87 TONS

DEVELOPMENT 926.0 FEET

MONTH OF November 1924

	Labor		Explosives		General Supplies		Power		Development		Mining		Development and Mining		Account
	Development	Mining	Development	Mining	Development	Mining	Development	Mining	Total	Per Foot	Total	Per Ton	Total	Per Ton	Number
Hand drilling—company acct	9.50	2,543.38	45.91	527.94	19.36	165.12			74 77	0 08	3,236.41	0 53	3,311 21	0 54	3,000
Hand drilling—contract	894.04	960 93	233.19	151.03	24.59	23.43			1,151.82	1 24	1,135 39	0 19	2,287.21	0 38	3,100
Machine drilling—company acct	231 51	3,230.22	221.05	2,349 28	123.81	678.22	178.35	1,889.11	751 72	0 81	8,146 83	1.34	8,901.55	1 46	3,200
Machine drilling—contract	3,682.37	17 38	1,927.36	25 84	228.23	206.46	1,214 24		7,052 20	7 62	219.68	0 03	7,301 88	1 20	3,300
Shaft sinking	522.13		24.23		556 99		39 31		1,112 66	1 23			1,142.66	0 19	3,400
Total	5,339 55	6,751.91	2,451 74	3,054 09	952 98	1,073.23	1,431.90	1,889 11	10,176.17	10 98	12,768 34	2 10	22,914 51	3 77	
Sharpening machine steel	357.32	513.79			145.15	406.69	62.30	129 58	564 77	0 61	1,080 06	0.18	1,644 83	0 27	3,500
Sharpening hand steel	214.25	220 45			3.10	12.40	5.66	5 66	223 01	0.24	238 51	0.04	461 52	0 08	3,550
Tool nipping	534 01	1,071.18			11 28	31.82	25.83	12 06	571.12	0.62	1,115 06	0 18	1,686.18	0 28	3,600
Drill pipe lines	502 04	568.25			329 34	753.88			831 38	0.90	1,322.13	0.22	2,153 51	0 35	3,700
Drill repairs and renewals	110 90	112 40			488 51	628.49	5 32	5.66	601 73	0 65	746 55	0.12	1,351 28	0 22	3,800
Total	1,718.52	2,516 07			977 38	1,833 28	99.11	152 96	2,795 01	3 02	4,502 31	0 74	7,297 32	1 20	
Timbering	1,722.65	6,274.40			1,494 62	4,304.06		27 75	3,217 27	3 47	10,606 21	1 74	13,823 48	2.27	4,000
Filling		673.18		19 03		16.51					708 72	0 12	708 72	0 12	4,100
Total	1,722.65	6,947.58		19 03	1,494 62	4,320.57		27 75	3,217 27	3 47	11,314.93	1 86	14,532 20	2 39	
Shoveling and sorting	227.20	3,604.81			4.79	297.84			231.99	0 25	3,902.65	0 64	4,134.61	0 68	4,200
Hoisting	48.85	1,710.87			51 29	235 36	34.83	104.49	134 97	0.15	2,050 72	0 34	2,185 69	0 36	4,300
General tramming	2,318 12	3,386 10			196.25	475 39			2,514.37	2.71	3,861.49	0 63	6,375 86	1 05	4,500
Tramming to mill		980 65				49.10		241.56			1,271 61	0 21	1,271 61	0 21	4,600
Total	2,594.17	9,682.43			252 33	1,057 99	34.83	346 05	2,881.33	3 11	11,096.47	1.82	13,967.80	2 30	
Sampling and assaying	169 91	1,196.58			31.32	348.72	6.94	93.12	208.17	0 22	1,638.42	0.27	1,846 59	0 30	4,700
Surveying	149.75	133.62			5 81	5 95			155.56	0 17	139 57	0 02	295 13	0 05	4,800
Clerical	521.57	1,669.77			10.12	38.17			531.69	0.57	1,707 94	0.28	2,239 63	0 37	4,900
Superintendence	1,088 50	2,473.63			31 22	32.04			1,119 72	1 21	2,505.67	0 12	3,625 39	0 60	5,000
Total	1,929.73	5,473 60			78.47	424.88	6.94	93.12	2,015.14	2.17	5,991 60	0 99	8,006.74	1.32	
Pumping	498.82	498.88			250.27	250 27	225.41	225 41	974.50	1 05	974.56	0 16	1,949 06	0 32	4,400
Diamond drilling	134 67				144.68		47.18		326 53	0.35			326.53	0.05	
Grand total	13,938.11	31,870.47	2,451 74	3,073.12	4,150.73	8,960.22	1,845 37	2,734 40	22,385 95		46,638 21	7 67	69,024 15	11.35	
Cost per ton	2.29	5.24	0.40	0.51	0.69	1.47	0 30	0.45	3.68		7.67		11.35		
Cost per foot	15.04		2 64		4.48		1 99			24.15					

Ore stoped 5,383 tons 8.66 cost per ton

Total hoisted 4,022 tons 0 54 cost per ton

Trammed to mill 6,409 ton miles 0.20 cost per ton mile

Shaft sinking ____ feet ____ cost per foot

Development and mining 0 64 tons per man-shift

Development direct 0.73 feet per man-shift

Machine drilling 20,065 feet 0.71 cost per foot

Drill sharpening 14,632 drills 0 112 cost per drill

Cost accountant

FIG. 5.—MONTHLY MINING COST STATEMENT.

thoroughly familiar with the operation of the mine, and he must exercise his judgment continually in distributing the various charges. Perhaps this close touch with changing conditions underground renders the report more valuable than would be possible were a more thoroughly systematized method employed. The total cost of drafting this financial statement of stopes is less than $50 U. S. currency per month. It is believed that this is returned many times through improved control of mining operations.

Costs

All costs at the mine are divided between development and mining. Development includes drifting and raising at 100-ft. intervals or more and maintenance of the passageways until stoping starts. Mining includes sublevels, intermediate raises, waste raises, stoping and maintenance after stoping has commenced. The costs of development and mining are further divided into labor, explosives, general supplies and power; and, under these headings are segregated the cost of the various occupations—drilling, timbering, shoveling, tramming, hoisting, superintendence, etc. The form of the cost sheet, with a record of typical monthly costs, is shown in Fig. 5. All important cost details are plotted monthly in order to show at a glance any unusual increases.

Safety

All possible steps are taken to prevent accidents. Detailed safety rules are posted about the mine and copies are given to new men. Besides the regular inspection of bosses and mechanics, special inspectors are chosen each month from among the best miners, to visit all working places and report to the mine superintendent. Bonuses are given to bosses who have had no serious accidents during the month. First-aid and helmet classes are held monthly.

DISCUSSION

Fred Hellmann,* New York, N. Y.—We used to take in all the raises weighted with their lengths on the dip. I think that the arithmetical average of all assays—if evenly spaced—could be used, but the method of weighting with the lengths would be more correct.

Benjamin F. Tillson,† Franklin, N. J.—When estimating the cubature to apply the values, the accepted engineering method is to use a prismoidal formula and calculate the volume on such a basis as to give a true volume on a regular figure. I would suggest the following method: Take the average area, or the area of each section, and for the block

* Consulting Mining Engineer, Guggenheim Brothers.

† Mining Engineer, New Jersey Zinc Co.

between two take the square root of the sums of the product, or the square of the first area plus the product of the two end areas plus the square of the other end areas; divide the sum of all these by three and extract the square root of the quotient. If you have figures and sections on which you can readily apply this method, it would be interesting to see how it checks out with the prismoidal formula, and whether it would not be more accurate when dealing with the geometric value of prismoids.

FRED HELLMANN.—Such accuracy is rarely needed in the calculation of orebodies. When figuring tonnage, you do not calculate by the prismoidal formula the contents of the triangular prism. It would be perfectly satisfactory to take the mean height and the mean cross section, and multiply the two. I cannot imagine any case where such accuracy would be demanded as to involve the use of the prismoidal formula.

BENJAMIN F. TILLSON—I take an interest in such accuracy because it is rather awkward to be called upon a witness stand and have the lawyer doubt the accuracy of the methods involved. It is valuable to him to point out some inaccuracy in scientific deduction.

Mining Methods of the Silver King Coalition

By Robert S. Lewis,* Salt Lake City, Utah

(New York Meeting, February, 1923)

Park City, Utah, elevation 7200 ft., is on the eastern slope of the Wasatch Mountains about 25 miles southeast of Salt Lake City, elevation 4200 ft. The town and surrounding mining district are served by two railroads: one, 28 miles long, is a branch of the Union Pacific R. R. and joins the main line at Echo; the other, 35 miles long, connects with the main line of the Denver & Rio Grande R. R. at Salt Lake City.

The main shaft and important mine buildings, elevation about 8200 ft., of the Silver King Coalition Mines Co. are 1½ miles by road to the southwest of Park City, and are near the head of Woodside Gulch. The property of this company is a consolidation of several groups of mining claims, and comprises an area that is roughly 4 miles long by 1 mile wide, the long axis of which has a N. 55° E. bearing.

Geology

A general geologic section through the property shows a basal formation of gray quartzite, called the Weber quartzite, above which seems to lie conformably the Park City formation. Both beds are of Carboniferous age. The Park City strata consist of a number of beds of limestone, with intercalated beds of quartzite, sandstone, and shale. The exact thickness is difficult to determine, but is apparently between 650 and 850 ft. Above the Park City beds and conformable with them lie the Woodside Shales. These thinly bedded fine-grained shales, about 1000 ft. in the thickness, are of no economic importance. They are overlain conformably by the Thaynes formation, which has about the same thickness and is composed of beds of limestone, sandstone, and shale. A thin bed of shale divides the formation into an upper and a lower part; the upper part contains more lime than the lower part. The Woodside shales and the Thaynes formation are of Triassic age. All the beds dip northwest, the dip is usually from 20° to 30°, but may be as great as 40°. In places, intrusions of diorite porphyry have deformed the beds. Marked faulting in zones of a northeast-southwest trend occurred after the intrusion of the porphyry. Later faulting in a

* Professor of mining, University of Utah.

northwest-southeast direction has made the task of working out the detailed geology of the ore deposits difficult.

Ore Occurrence

The lode deposits are found in the fissures which were also channels through which the solutions circulated and formed the replacement deposits in the purer limestone beds. These replacement deposits vary greatly in width and range from the full thickness of a limestone bed to only a small fraction of this width. The hanging wall of these deposits is usually good, being sandstone in nearly every case, and stopes that are not too large may be left unfilled for a long time. The lowest bedded deposit has a strong foot wall in the quartzite, but upper beds have a limestone foot wall, which may not be good.

Because of the flat dip of the beds and the flatter slope of some of the oreshoots which do not extend directly down the dip but quarter across it, the ore does not flow readily through wooden chutes, but must be worked down with rakes. The ore, down to the 900-ft. level, is nearly all carbonate. Below this level is a sulfide zone 150 to 200 ft. thick, below which carbonate ore is again found. From the 1300-ft. level down, all the ore is sulfide. The sulfide ore is harder than the carbonate ore.

Many horses as well as bunches of low-grade ore are found in the rich ore. There seems to be no regularity in their occurrence, and they may be found anywhere in the orebodies. The waste ranges from 5 to 50 per cent. of the ore. The Woodside shales hold much water, some of which works down into the Park City beds. The contact of the Park City formation and Weber quartzite is usually dry.

The silver King oreshoot, though almost continuous, subdivides in places and the branches extend irregularly in the same general direction. The Silver Hill bedded deposits are one above the other, with barren beds between. These deposits extend down the northerly dip of the beds for not more than 100 to 150 ft. from the Crescent fissure; they become thinner toward the lower edge. Their long axis is parallel with the fissure and may be several hundred feet in length. The fissure dips to the south and the ore is found only on the foot-wall side of the fissure.

Ores

The ore contains lead and silver with minor quantities of copper and gold and from 2 to 30 per cent. zinc. It is found both as lode deposits in the fissure zones and as replacement deposits in the purer limestone beds. The sulfide ore is mainly galena with intergrown tetrahedrite and pyrite. Some anglesite, sphalerite, cerusite, and a little malachite may be present. The ore in the upper levels has been oxidized, but the zone of oxidation is irregular. Oxidized ore has been found below beds that contained nearly pure sulfide ore.

The ore is sorted underground into first-class ore, which is shipped to the smelter and second-class or milling ore. The first-class ore averages 0.068 oz. gold, 45 oz. silver, 27 per cent. lead and 0.72 per cent. copper. The milling ore assays from 4 to 7 oz. silver and 5 to 8 per cent. lead; the average grade of the concentrate is 0.045 oz. gold, 26 oz. silver, 34 per cent. lead and 9.94 per cent. copper.

Development

The chief mine openings are the Alliance tunnel and the Silver King shaft. The collar of this shaft is in the Park City formation just below the contact with the Woodside shales. All ore mined is hoisted to the surface through this shaft. Near the shaft are the mine office, boarding house, shops, tramway loading station, sampler, and mill. A small quantity of timber and supplies are brought into the mine through the Alliance tunnel, the portal of which is some 4600 ft. from the Silver King shaft and about 9500 ft. from the Silver Hill shaft, which was sunk from a point on the Alliance tunnel. The station, at the collar of this shaft, is connected with the 500-ft. level of the Silver King shaft.

Practically all of the ore that was near the surface has been mined, so that production is limited to the lower levels of the mine. However, the southwest part of the company's property has not yet been developed. Fig. 1 shows the principal mine workings projected into the planes of the different vertical sections. A large bedded deposit, which is nearly continuous, dips northwest from the Silver King shaft near the 700-ft. level. The 1300-ft. level was extended to cut the deposit at greater depth. Ore was found in this level at a point about 3500 ft. from the shaft. As the distance was too great to warrant driving a still lower level from this shaft to reach the orebody, the Central shaft was sunk in the foot wall of the deposit, from the 1300-ft. level to the 1450-ft. level. Further development of the orebody is carried out through this shaft. The foot wall is the Weber quartzite, and little ore is found at a greater distance than 200 ft. above the contact of the Weber quartzite and the Park City formation. The "M. L. M." oreshoot, as it is called, is worked on the southern edge below the 1300-ft. level from the Central shaft, but on the northern edge through inclines driven from the 1300-ft. level of the Silver King shaft.

The Silver Hill shaft was sunk from the station at the inner end of the Alliance tunnel, which is 9500 ft. long and is connected to the 500-ft. level of the Silver King shaft by a crosscut 2200 ft. long. The Silver Hill shaft was to be used in mining the deeper extension of the orebodies found in the Silver King and Central shafts, and would also serve in the developing of the company's property to the southwest. The ore deposits along the Crescent fissure could be reached through a crosscut to the north, which would be well below the old workings in this fissure.

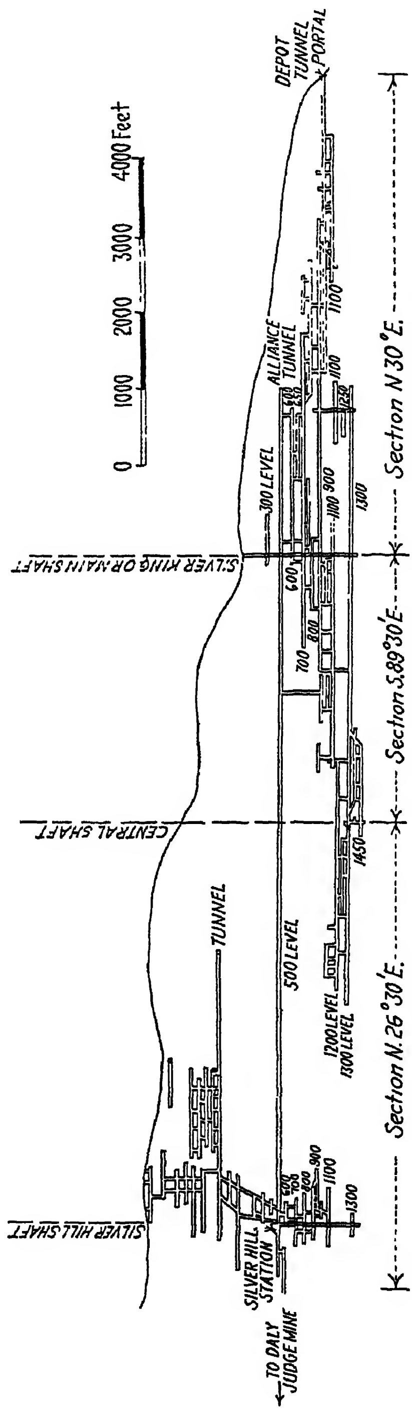

FIG. 1.—LONGITUDINAL SECTION OF WORKINGS OF SILVER KING COALITION MINE.

These old workings were in the Thaynes formation, but this new development would all be in the Park City formation. As the Silver Hill station is 1650 ft. vertically below the surface, it was thought best to have the station underground rather than to drive the shaft up to the surface at a point from which it would be difficult to get the ore down to the tramway or mill, a thousand feet or more below. All ore from the Silver Hill shaft is hauled through the Alliance tunnel and crosscut to the 500-ft. station of the Silver King shaft, from which point it is hoisted to the surface.

Mining Method

The general method of mining is overhand stoping. The main levels are 100 ft. apart, but sublevels are driven wherever they can be advan-

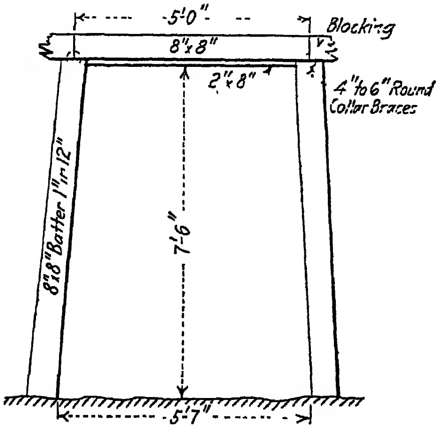

Fig. 2.—Standard level or drift set.

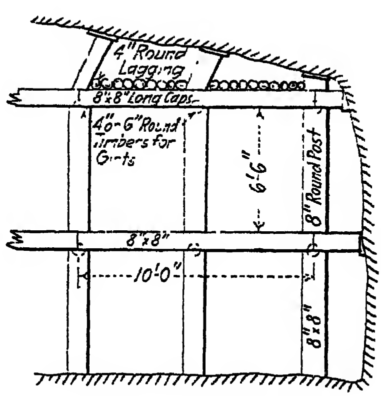

Fig. 3.—Stope set.

tageously used. In case the ore extends for only a short distance below a main level, as is learned by prospecting with small winzes, the ore is worked by underhand stoping from the level. If the ore continues to a much greater depth, a lower level is driven and an overhand stope is started.

Levels and main drifts are 5 by 8 ft.; the standard level or drift set is shown in Fig. 2. The sets are 5 ft. center to center. The smaller ore-bodies are worked through winding drifts, less than 5 by 8 ft. in size, which are driven in the ore. In some places wheelbarrows are used to carry ore a short distance to the chutes. In other stopes, mine cars are used. There are a few stopes in which the ore is transferred from the bottom of a chute, which lies on the foot wall at so flat a slope that the ore must be worked down the chute, to a car in a sublevel and trammed to another chute that connects with the level below. In the Silver Hill section, there is no connection to the shaft from the 700 to the 900-ft. levels. All ore mined between these levels must be dropped through

chutes to the 900-ft. level. The shaft was sunk to the 1300-ft. level, but it filled with water to the 950-ft. level during a strike a few years ago, and it will not be unwatered until there is need to go below this level for ore. The stopes between the 900 and the 950-ft. levels are served by inclines on which a small compressed-air hoist handles one car at a time.

In the small stopes, posts are used to support the roof. They are put in at irregular intervals as needed. In the medium-size stopes square sets are used; a set is shown in Fig. 3. The 8 by 8 in. timbers are Oregon pine; the round timbers are common red pine. The sets are 5 ft. square, but a 10-ft. cap is used to tie the sets together. (Girts are small round timbers that are either spiked to the posts and caps or else to small blocks that are nailed to the posts.) The posts are unframed. No sills are used unless the floor of the stope is ore that is to be mined from the level below. In large stopes and where the ground is hard to hold, a

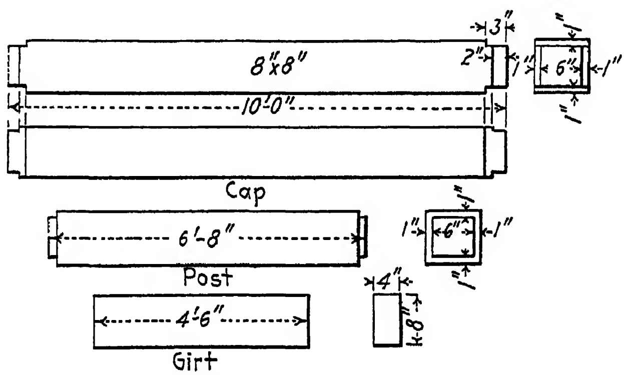

FIG. 4.—TIMBER USED IN STOPE SETS.

different framing is used; only the girts are unframed, as is shown in Fig. 4.

In stopes that are several sets high, the ore is dropped to the sill floor either through wooden chutes or else through 12-in. galvanized pipes, called "sucker pipes." Only the sloping bottom of a chute is placed in the upper part of a sill-floor set and two or three of these pipes radiate upward from this chute bottom to different parts of the stope.

SAMPLING AND SORTING UNDERGROUND

Most samples are grab samples taken from the cars underground, though a few grab samples are taken from the cars when they reach the surface. They are taken according to the judgment of the sampler. The assays returns from these samples are from 5 to 10 per cent. higher than those from the sampling mill. Newly opened stopes are given particular attention for a few days. Groove samples, taken with a pick, are cut daily. It is seldom necessary to sample the first-class ore,

so most of the samples are of second-class or milling ore. The ore is sorted underground. The first-class ore is picked out by hand or else all the ore is screened through a 1-in. screen, and the large pieces of high-grade ore are picked out to go with the fines, which are usually rich.

Stope Filling

About half of the stopes are filled. Small stopes that will soon be worked out or that will not cave are seldom filled except where accumulated waste must be disposed of. Whether or not a stope should be filled depends on the size of the stope and the likelihood of its remaining intact. The filling may be largely the waste in the ore or it may be dropped through chutes from levels above; or a waste drift may be driven into the wall near the top of the stope and the waste dumped directly into the stope from a track over the upper sets.

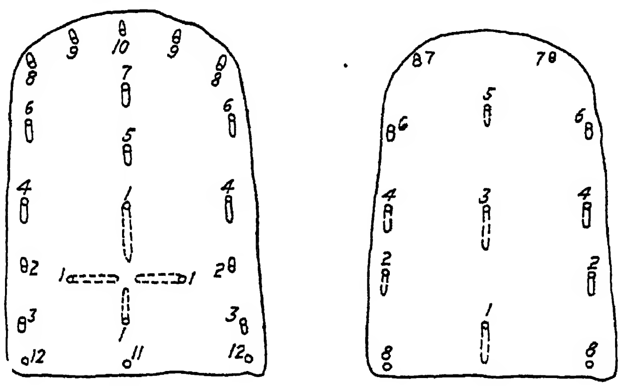

Fig. 5.—*A*, Drill round in drifts of medium hardness. *B*, Drill round in quartzite and hardest ground.

Large chutes convey the ore from one level to another. These chutes have three compartments, each being 40 by 52 in. inside the 8 by 8-in. timbers. The two outside compartments are ore chutes; the middle compartment is a manway. Two-inch lagging is placed lengthwise of the chute inside the timbers, thus making the compartments 3 by 4 ft. in the clear. The sets are placed 5 ft. center to center. Chute gates are 2-in. boards, that slide between 4-in. strips nailed to the sides of the chutes. Steel gates, operated by a lever, were tried but did not prove satisfactory. Timbers are framed on the surface; only unframed posts are cut underground and then only on special occasions. All timbers are brought into the mine at night and are distributed to points near the working places.

Blasting

Hercules gelatin dynamite is used in blasting. For machine-drilled holes 1⅛-in. sticks are used; but ⅞-in. sticks are used in hand-drilled

holes. In soft ground, 35 per cent. dynamite is used; but the 50 per cent. strength gives better results in hard ground. Victor fuse and No. 8 caps are used. The priming stick is the last one put into the hole. The charge is not tamped.

In ground of medium hardness thirteen holes form a round for drifts; see Fig. 5*B*. For 6-ft. holes the advance is from 5 to 5½ ft. per round. In the quartzite and hardest ground, the round consists of twenty-two holes placed as shown in Fig. 5*B*. About 4-ft. advance is made with 5-ft. holes; 50 lb. of explosive is used in each round. In the softest ground as few as seven holes may be required for a round. In stoping, as few holes as possible are used. A single hole may break the full height of the ore in the thin-bedded deposits.

The drills used are as follows: Ingersoll-Rand Leyner, for drifting; Jackhamer drills, both wet and dry, for stoping; small stopers, Ingersoll

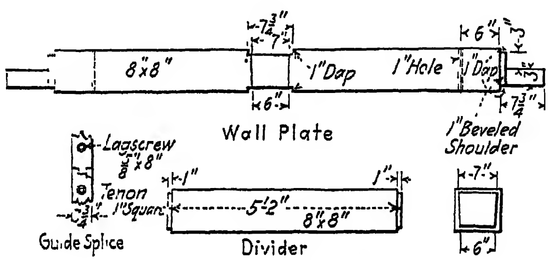

Fig. 6.—Shaft timbers.

CC11 dry and CC11W, wet; Waugh stopers, 16V and 16 VW; and self-rotating Denver rock drill No. 75.

The drill steel used is: for Leyner drills, 1⅛-in. round hollow steel with cross bit; for wet stopers, ⅞-in. hexagon hollow steel with cross bit; for dry stopers, 1-in. cruciform steel with cross bit; and for Jackhamer drills, ⅞-in. hexagon hollow steel, with collar and cross bit.

Shafts

The Silver King shaft has three compartments; two hoisting compartments 4½ by 5 ft. in the clear and a manway 2½ by 5 ft. The timbers are 10 by 10 in., and the sets are 5 ft. center to center. The Silver Hill shaft also has three compartments. The two hoisting compartments are 4½ by 5 ft., the other compartment is 3 ft. 8 in. by 5 ft. It is fitted with a small cage 2½ by 4 ft. in size and is operated by a hoist using compressed air. Three 6-in. pipes are also in this compartment; two are for water and the other is for compressed air. The Central shaft has two compartments 4½ by 5 ft. The timbers are 8 by 8 in., and the sets are 5 ft. center to center; Fig. 6 shows the method of framing. The end plate is framed in the same way as the wall plate; the posts are unframed.

All timbers are framed from one side to allow for any variation in size of the sticks. The guides are of selected Oregon pine 4 by 5 in. planed down to 3¾ by 4¾ in. They are 20 ft. long and span four sets. A tenon 1 in. square is used at the splices. One ⅝ by 8-in. lag screw, countersunk, is used at the end of each guide and at every shaft set.

Underground Haulage

The mine cars are 24 in. deep, 44 in. long, and 28 in. wide; they hold 17 cu. ft. The track gage is 18 in. The cars are fitted with McCaskell wheels 12 in. in diameter, but all new cars will have ball bearings. The trucks have a framework of oak beams, and are fitted with a turntable. The cars weigh about 1100 lb. each; they are end-dump and have a coiled spiral spring on the solid end to act as a buffer. They hold about 1800 lb. of second-class ore and about 2160 lb. of first-class sulfide ore. Where hand tramming or animal haulage is used, the track is of 12 or 16-lb. rails. One horse and one mule do the hauling on the 1450-ft. level of the Central shaft. Motor haulage is used on the 900-ft., 1100-ft. and 1300-ft. levels of the Silver King shaft, and for hauling the ore from the Silver Hill station to the 500-ft. level of the Silver King shaft. One motor on the surface serves the sampling mill and the concentrator. The rails for the motor track range in weight from 20 to 40 lb. but all new track will have 30-lb. rails as the standard. Two types of motors are in use: one is the 4-ton Westinghouse bar-steel frame, with steel-tired wheels and 250-volt ball-bearing motors, with a rated speed of 4.7 miles per hour; the other is the 3-ton General Electric type L. M. 2T3, A-1, with two 250-volt motors; drawbar pull on level track is 1200 lb., at rated speeds of 5.2 and 2.7 miles per hour. No provision is made for underground storage of ore, except as described under the heading of Silver Hill station. However, the shaft stations have sufficient double track to accommodate a large number of cars.

Hoisting

The hoist at the Silver King shaft is a Bullock-Corliss steam hoist of 500 hp. Each of the two reels is controlled by a clutch and can be operated independently of the other. The flat rope is ⅜ by 6 in., and is made of seventeen strands of crucible-steel wire. Each strand is made up of four smaller strands of seven No. 16 wires. The strands are alternately right and left and are sewed together by No. 16 soft-iron wire.[1] Hoisting is done in balance, with double-deck cages that weigh 2400 lb. each. During uninterrupted hoisting, as many as 60 cars an hour have

[1] All the flat ropes are made at the mine; two men can finish about 50 ft. a day.

been handled from the 1300-ft. level, but this is unusually continuous hoisting. The average hoisting period is 50 sec.; this includes acceleration and retardation. About 35 sec. is required at each landing to change cars, so that the total time of the hoisting cycle is about 2 min. Landing chairs are used in the shaft.

The Silver Hill station has a 38½ ft. steel head frame set on concrete piers. The center of the sheave wheels is 46.5 ft. above the floor of the station. The hoist room is 35 by 50 ft. and has a maximum height of 50 ft. The adjoining compressor room is 25 by 44 ft. and is 19 ft. high. Both rooms have concrete walls, which range in thickness from 2 to 5 ft. The 400-hp. hoist was made by the Wellman-Seaver-Morgan Co. Direct current at 600 volts is supplied from a motor-generator set of 165-kw. capacity. Alternating current is transmitted at 2200 volts. The reels are 3½ ft. inside diameter and 10½ ft. outside diameter. The skips hold 52 cu. ft. or about 3 tons. The ore is hoisted from loading pockets on the 600-ft. 700-ft. and 900-ft. levels, and is dumped into a timber bin of 75 tons capacity. Each of the two bin doors is controlled by a rod that connects with the piston of a compressed-air cylinder fastened to the bin above the door. The Central shaft is equipped with an electric hoist of 200 hp. A ⅝-in. round rope and cylindrical drum are used at this shaft.

Drainage and Pumping

At the Silver Hill shaft, the pump station is 50 ft. below the 900-ft. level. The pump is a belt-driven, single-acting, Deane, triplex pump, with 7½ by 12-in. cylinders. It lifts 350 gal. per min. against a 450-ft. head, and runs 18 hr. a day. The discharge pipe is 6 in. in diameter. This water runs out of the mine through a ditch 4 ft. wide and 2 ft. deep cut in the bottom of the Alliance tunnel.

At the Silver King shaft, the pumps are at the 1300-ft. level. One pump is a No. 5 Cameron pump of 50 gal. per min. capacity. It requires 25 hp. for driving but is used only in an emergency. The other is a 25-hp. Gould triplex pump, with 4½ by 8-in. cylinders. Its capacity is 80 gal. per min. This pump lifts the water only 8 ft., just high enough for it to run into some old stopes through which it finds its way out of the mine.

Air Compression

An air compressor is used at each shaft; the pressure is from 80 to 100 lb. per sq. in. at the compressor. All air lines are connected together underground. Some details are as follows:

Shaft	Type	Size, In.	Capacity, Cu. Ft. Free Air at Sea Level	How Driven
Central..	Sullivan angle	18 by 11 by 14	950	Direct-connected to 175-hp. induction motor.
Silver King......	Chicago Pneumatic	20 by 11 by 12	1065	Belted to 200-hp. synchronous motor.
Silver Hill.......	Sullivan angle	18 by 11 by 14	950	Belted to 175-hp. synchronous motor.
Silver Hill... ..	Sullivan angle	20 by 11 by 14	1065	Direct-connected to 175-hp. synchronous motor.

Ventilation

Because of the numerous connections between levels and the several openings to the surface at different elevations, nearly all working places are adequately served by natural ventilation, aided by a few doors in the drifts. Three small fans of the Sturtevant propeller fan type with self-contained motors ventilate the lower levels. Two fans are 42 in. in diameter and have a 1-hp. motor; the other fan is 48 in. in diameter and has a 2-hp. motor. They can supply air at only a few ounces pressure, but that is sufficient to maintain a circulation of air in the bottom stopes and levels. The air and rock temperatures are low. Observations that the writer made in December, 1922, when the air temperature outdoors was 26° F., were as follows: At a point well inside the Alliance tunnel, 46° F.; in the Silver Hill station about half way between the hoist motors and the shaft, the temperature near the floor of the station was 57° F.; on the 800-ft. level of the Silver Hill shaft 53° F.; on the 900-ft. level and on the sill floor of the large stope on the 950-ft. level 50° and 52° F. respectively.

Lighting and Signaling

All shaft stations are electrically lighted. A few sharp curves in the Alliance tunnel and also in the crosscut to the Silver King shaft are lighted by electricity. The miners use carbide lamps, nearly all of which are the small size.

The main shaft stations are connected to the mine office by telephone. Electric flash signals are used to call the cage at the shaft stations. A bell rope, operated by levers close to the shaft, is used to signal the engineman to hoist.

Sampling and Shipping Ore

All of the first-class shipping ore and the concentrate from the mill are put through the sampling mill before being sent over the tramway to the railroad. In this way, an accurate sample of the material shipped to the smelters is obtained. The sampling mill is equipped with several

sets of rolls and Vezin automatic samplers. The sampling is accurately done.

Aerial Tram

The tramway is about 7000 ft. long. In profile, it rises about 200 ft. a short distance from the sampling mill to surmount the crest of a ridge and then drops about 1200 ft. to the terminal at the railroad. Eighty buckets are in use; they are spaced 175 ft. apart. Each bucket has a capacity of 500 lb. of ore or 325 lb. of coal. Coal for the mine is brought over the tramway as are many of the mine supplies. The speed of the buckets is 150 ft. per min. The steel towers are made of 4 by 4-in. angle irons, and range in height from 16 to 65 ft. The longest span is 200 ft. The traction rope is ¾ in. in diameter, and the standing ropes are 1⅛ in. on the ore side and 1 in. on the return side. These ropes are of crucible steel, which has a breaking strength of 224,000 lb. per sq. in.

Operating Data

The data following is for the period April to November, 1922, inclusive. So little contract work is done, compared with day labor, that all work may be considered as day labor.

Ore Production

Work by]	Tons per Man-hour	Man-hours per Ton
All miners	0.52	1.92
All underground labor	0.215	4.65
Total organization	0.113	8.84

Classification of All Employees

Class	Per Cent.	Class	Per Cent.
Mine	75.2	Shops	7.3
Sampler	1.5	Stovehouse	0.2
Mill	10.6	General	2.4
Tramway	2.4	Office	0.4

The nationality of employees is 63 per cent. American, 30 per cent. Irish, and the others mostly Finns and Slavs. The labor turnover, calculated on a yearly basis, is 390 per cent.

Of the total mining costs, 68 per cent. goes to labor and 32 per cent. for supplies.

Supplies and Power

Explosives:	
In drifts, pound per foot	11.
Ore extraction, pound per ton	0.82
Development and extraction, pound per ton	1.86
Timber and logging (11 months):	
Development, boardfeet per foot	6.9
Ore extraction, boardfeet per foot	3.22
Total per ton extracted, boardfeet	3 93
Power:	
Total required, horsepower	1400.
Mill, horsepower	188.
Mine and surface, horsepower	1212.
Air compression, horsepower-month per ton of ore	0.09
Hauling and hoisting, horsepower-hour per ton	10.
Pumping, horsepower-hour per ton	10.
Lighting, horsepower-hour per ton	0.3

Compressed air consumed per ton of ore amounts to 27.2 cu. ft.

Acknowledgment

The author wishes to acknowledge his indebtedness to J. M. Dailey, general manager of the Silver King Coalition, for his courteous permission to visit the mine and look through the records, and also for his assistance in collecting data.

Shrinkage Stopes

A SHRINKAGE stope is an overhand stope in which the broken ore accumulates until the stope is completed to, or near, the level above. As broken ore generally occupies at least 60 per cent. more space than ore in place, about a third of this must be drawn out as the stope advances to leave a working space under the back. The ore remaining supports the miners and gives temporary support to walls, and is all withdrawn when the stope is finished. The shrinkage-stope method is applicable to steep-dipping deposits of strong ore with strong walls. It may be applied to narrow or wide veins, and is simple and cheap. Five examples, which include important and varied types of deposits, are given: namely, Kennecott, Homestake, Jarbidge district, Mogollon district, and Cripple Creek district.

Geology and Mining Methods of Kennecott Mines

By Stephen Birch, New York, N. Y.

(New York Meeting, February, 1924)

The Chitina mining district of Alaska is located at the headwaters of the Chitina and Copper Rivers. At present, the only producing mining properties are the mines of the Kennecott Copper Corpn. and the Mother Lode Coalition Co., which are situated 196 miles from Cordova, the port of entry. The first claims, later acquired by the Kennecott Mines Co. and afterwards transferred to the Kennecott Copper Corpn., were discovered in 1900. The Copper River & Northwestern Ry., which connects the mines with tide water at Cordova, was completed in the spring of 1911.

Contemporary with the construction of the railroad, aerial tram equipment was brought to the mines by pack train and a tramway, 3 miles long, connecting Bonanza mine with the proposed railroad terminal, was finished, enabling shipments of high-grade ore to be made immediately the railroad was completed. A mill to treat the lower grade ore was begun the same year.

The Kennecott company's holdings consist of 111 mineral claims. The Mother Lode Coalition Mines Co., which is controlled by the Kennecott Copper Corpn., owns 73 claims adjoining the Kennecott holdings. All data on operations and geology refer equally well to the Mother Lode property.

Geology

The general geology of the district has been covered by the U. S. Geological Survey[1] and the geological features of the mines have been carefully studied by A. M. Bateman, in his capacity as consulting geologist to the company.[2]

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[1] Fred H. Moffitt: U. S. Geol. Surv. *Bull.* 662 (1918) 164; Fred H. Moffitt and S. R Capps: U. S. Geol. Surv. *Bull.* 448 (1911).

[2] A. M. Bateman and D. H. McLaughlin: *Econ. Geol.* (1920) **15.**

The formations in the vicinity of Kennecott are shown, by the U. S. Geological Survey, to be as follows:

Quaternary.—Alluvium: flood plain gravels, sands and silts.
Rock glaciers: broken rock and ice.
Moraines: glacial till, partly sorted.

Jurassic or later.—Quartz diorite porphyry: stocks, sills, and dikes.

Upper Jurassic.—Kennecott formation: shales, sandstones, and conglomerates.

Upper Triassic.—McCarthy shale: shale with few thin-bedded limestones.
Chitistone limestone: massive limestone mostly magnesian, ore containing.

Triassic—Nikolai greenstone: altered basaltic lava flows.

The Nikolai greenstone is a succession of altered basaltic lava flows; its total thickness, exposed in the vicinity of the mines, is at least 3500 ft. and the base cannot be seen. Numerous prospects have been opened on copper showings in this formation, the ore being usually bornite, chalcopyrite, and occasionally chalcocite; however, they have not resulted in productive mines. Native copper is known to occur in all placer operations in gulches cutting the greenstone; some of the nuggets weigh several hundred pounds. In the vicinity of the mines, the strike of the greenstone is N 60° W and its dip 23° to 30° to the northeast.

Chitistone Limestone.—All the important orebodies are in this formation. It is a conspicuous heavy-bedded formation intersected by numerous systems of fracturing; weathering along these fracture planes produced a very rugged topography. It conformably overlies the Nikolai greenstone and is estimated, by Moffitt, to be about 3000 ft. thick.

The lower part of the formation consists of a 4–7-ft. bed of shale; above the shale is 12 ft. of thin bedded, smooth, hard, gray argillaceous limestone, then 23 ft. of thin-bedded, rough, pebbly limestone, containing flattened, cylindrical, fossil-like grains which, from its appearance, Bateman has termed "crinkley lime," and 30 ft. or more of dull gray limestone. The remainder of the formation consists of massive beds of sparkling light-gray dolomitic limestone, with occasional beds of darker rock. The upper part of the Chitistone limestone becomes thinner bedded and shaly, gradually grading into the overlying McCarthy shales.

Porphyries.—Light-colored quartz diorite porphyries intrude the greenstone and all the sedimentary rocks in the form of stocks, sills, and dikes. They occur most abundantly about one mile from the Bonanza mine, where they form a larger stock, which constitutes Porphyry Mountain.

Faults and Fractures

There are numerous faults both parallel to and traversing the bedding of the sedimentaries. The former are known as flat faults; the latter also

pass into and displace the greenstone. There are many displacements of from 1 to 25 ft., and several faults caused a displacement of as much as 1300 ft. Most of these were pre-mineral; however, in the Bonanza and Mother Lode mines there are several instances where a portion of the orebody has been displaced. Bateman considers that the flat faults have had a direct bearing on the deposition of the ore, the selvage or gouge contained in them acting as a dam to the orebearing solutions.

Ore Deposits

The general geological features and the relative position of the mines are shown in Fig. 1. The orebodies are typical replacement deposits in the limestone, the outstanding features being the intensity of the mineralization and the fact that chalcocite is the predominating mineral in the deposits. As usual, deposition took place along a fissure, or series of fissures that seemingly start from the greenstone contact.

These fissures have a strike varying from N 30° E to N 80° E and have no definite dip, varying from nearly vertical to 40° from the vertical; most of them more closely approach the vertical, however. The orebodies have the same strike and dip as the fissures, although often when a fault plane is intersected, they widen out along these planes and form what are termed the "flat orebodies," and are identical with the "Manta" orebodies of the Mexicans. The mineralization along the fissures is much less as the fissure passes into the dull gray limestone, and in only two or three instances is any ore found in this formation or the "crinkley lime" beds that immediately overlie the greenstone.

In the Jumbo mine, a fault roughly following the contact between the dolomitic and the dull gray limestone is the west limit of an orebody, the largest mass of high-grade ore so far encountered. This deposit had a cross-section of 80 by 100 ft. and extended from the 150-ft. to the 700-ft. levels, of which a portion 50 ft. wide and 50 ft. high, extending from the 300-ft. to the 600-ft. level, was practically pure chalcocite.

The lower 1000 ft. of the dolomitic limestone appears to be the most favorable zone for ore deposition. All the productive orebodies lie in it and have their greatest width in the lowest beds, gradually becoming smaller and of lower grade as they extend east into the upper beds. The eastern extension of the fissure is usually filled with calcite. Thus, the orebodies have a rake or pitch practically paralleling the greenstone contact.

There is every degree of intensity of replacement, from large bodies of practically pure chalcocite and its oxidation products, covellite, azurite, and malachite, to the lime containing small bunches or veinlets of these minerals too low grade to mine. There are no defined walls; the grade of the ore is the limiting factor in mining.

In width, the orebodies vary from a few feet to over 100 ft., not including the local widening of the flat orebodies, which sometimes extend another 100 ft.; in length they vary from 150 to over 1000 ft. In some places, practically the entire width is high-grade ore with only a few feet

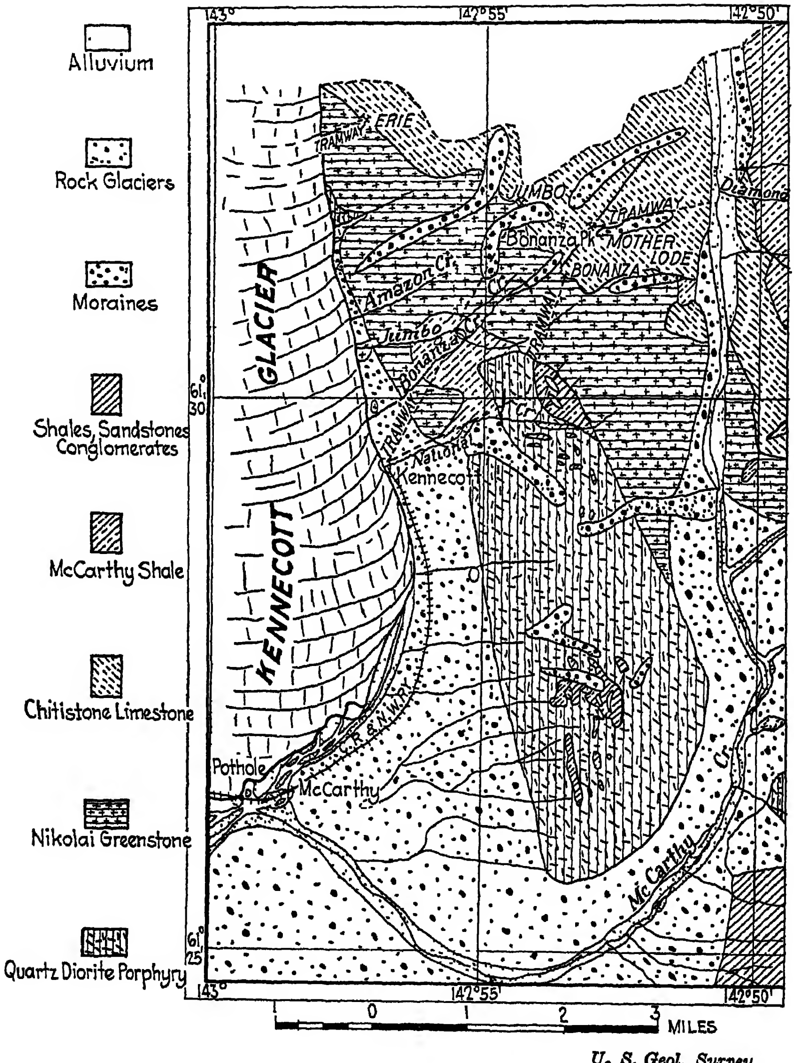

FIG. 1.—GEOLOGY IN VICINITY OF KENNECOTT.

of lower grade; in others, the high-grade is in veins from 1 to 10 ft. in width, which are separated from one another by lower grade ore. As the eastern ends of the orebodies are reached, with but one exception, no high-grade deposits are found. There are several places where it would appear that pre-existent fissures or veins were filled, but this occurrence is rare.

The Glacier mine exploits a unique and interesting orebody. It is made up of ice, limestone, some greenstone, and chalcocite. The outcrop of the Bonanza mine was a massive deposit of chalcocite located on the edge of a small amphitheater; the debris, resulting from disintegration of this orebody and country rock, fell into this basin and was occluded in a glacier, which now partly fills it. The orebody is 800 ft. long and 85 ft. wide, and the broken ore in payable quantities extends to a depth of 40 ft.; 45 per cent. of the volume is ice, the remainder is broken country rock and chalcocite with a small amount of carbonate ore.

The principal mineral is chalcocite and its oxidation products covellite, malachite, and azurite. Enargite, bornite, and chalcopyrite are occasionally found together with cuprite, luzonite, and other rarer copper-bearing minerals. During the past five years, the ore produced has averaged 70 per cent. sulfides and 30 per cent. carbonates. The ore is divided in two grades: that which is shipped direct to the smelter and the lower grade ores, which are treated in the mill and leaching plant. The high-grade shipments average between 50 and 55 per cent. copper.

Silver exists in the ore in the ratio of about 1 oz. silver to each 130 lb. copper.

General Description

The Jumbo and Bonanza mines are located on the greenstone-limestone contact at an elevation of 6000 ft.; the Erie mine, on the same contact, is at an elevation of 4500 ft.; and the Mother Lode mine is at an elevation of 5200 ft. This last mine was opened in the higher beds of limestone, the vertical shaft intersecting the contact at an elevation of 4400 ft. Contrary to all expectations, the temperature at the elevation of the mine is not extremely cold, rarely falling below $-20°$ F. and during the winter is often 40° warmer than at the mill camp 4000 ft. lower. Freezing or near freezing temperatures prevail even at the lowest levels of the mines, so the mines are dry and dusty; veins of ice are commonly encountered. The only pumping required is during the summer months, when the snow melts and a small part of the water finds its way through open fissures to the upper levels.

The topography is extremely rough and rugged; snow lies on the ground nine months of the year and snow falls throughout the year. Because of the topography, space for bunk houses and other buildings is limited. All hoists, compressors, and other machinery are located underground. Aerial tramways transport the ore to the mill or railroad terminal, all supplies to mines, and, during the winter months, carry all the passengers to and from the mines.

All the mines, except the Erie, are connected underground; a tunnel is now being driven to connect this mine with Jumbo. Jumbo and Bonanza mines are opened by inclined shafts paralleling the dip of the

greenstone and are located about 50 ft. above the contact. These shafts are 14 ft. wide, have two skipways and a manway, and are 7 ft. high above the rail. The shaft of the Jumbo mine has a slope distance of 3051 ft. and the shaft of the Bonanza 2416 ft. On account of the flat dip, the manways have stairways in place of ladders.

The skips used at Jumbo have a capacity of 80 cu. ft. and those at Bonanza, 60 cu. ft., with a track gage of 40 in. in both shafts. The Mother Lode mine was opened by a two-compartment vertical shaft 800 ft. deep. A new incline shaft has been sunk a slope distance of 1405 ft., after the same manner as at the other mines. All are located underground, being connected with the surface by a tunnel. On account of the flat pitch of the orebodies, the vertical shafts would require an excessive amount of development work to open the various levels.

Formerly, levels were driven each hundred feet, this distance was increased to 200 ft., which was found to be too great, and 150 ft. has been accepted as the best distance, all things considered. Two or three pockets are commonly cut at each level and the skips loaded by chutes without a measuring hopper. One pocket for the mill ore is usually capable of holding about 300 tons; the others, for the high-grade and waste, have a capacity of 50 to 100 tons.

Exploration, Sampling, and Estimating

In common with most deposits in the limestone, it is impossible to foretell or estimate accurately the amount or grade of the ore that a block of ground will produce without an unreasonable amount of development work. Diamond drilling has been used to good advantage for exploring unknown ground; in all over 70,000 ft. of drilling has been done. The usual and more reliable method of exploring has been to drive a drift or crosscut in the dolomitic limestone paralleling the strike of the greenstone, and about 100 to 150 ft. from it; thus any mineral-bearing fissure that is encountered can be followed.

Only occasionally is any sampling done underground. After becoming acquainted with the ore, it is possible to estimate closely the grade of the ore by the amount of glance or carbonates it contains. When the limits of the ore are reached, samples are sometimes taken. It has been found that the sample values are usually considerably higher than the actual recovery obtained in the mill; this is probably due to the friability of the glance and the soft chalky nature of some of the carbonates.

Mining Methods

The shrinkage method of stoping has been used, except for the open-pit mining on the Bonanza mine outcrop. A departure from the usual method, however, is practiced. Where the high-grade portion of the orebody is of sufficient size, as much as possible is mined by the shrinkage

method and completely drawn out. The mill-grade ore is then stoped, filling the void left by the extraction of the high-grade ore, and the excess is drawn off as usual.

After as much of the high-grade ore is mined as is practical, other veins, lenses, and masses are met and broken with the mill ore. No attempt is made to sort the ore in the stopes after the mining of the mill ore is commenced; but at all the mines, the ore from the skip pocket on the top level passes over a picking belt, where pieces of high-grade ore are hand picked from the mill ore and any mill ore that may be mixed with the high-grade produce is picked out.

The character of the ground makes almost an ideal condition for the method employed. The work must be given close attention to guard against leaving ore that makes along bedding planes, faults or cross fissures, away from the main orebody. Many of the floor pillars left are recovered after a level is finished; but it has been found that it is well not to be too hasty about the recovery of pillars and destroying the level, as oreshoots from a lower level have been found in ground that was considered barren. Until recently, no attempt was made to fill these old stopes, as they would stand empty with practically no caving; the waste from development work is now being used for this purpose.

The Glacier mine is worked but three months per year, when surface mining is carried on. During July, August, and September, the ice of the glacier melts sufficiently to release about 30,000 tons of ore; this is recovered by scraping the thawed ground with a Bagley scraper. To date, while some experimental work has been done, thawing by artificial means has not been attempted; possibly operations might be successfully carried on during the cold months, but it would be at a much greater cost. The scraper used has a capacity of 50 cu. ft. and is operated by an electric double-drum engine of 75 horsepower.

Development Plans

As the inclined shafts are located on the western limits of the ore, crosscuts are driven until the orebodies are reached. The drifts on the ore are kept, as far as possible, in the high-grade ore; chute raises are driven 25 to 35 ft. apart, and widened in the usual manner so that they connect, leaving a pillar 25 to 30 ft. thick between the level and the bottom of the stope. Often, if the ore becomes leaner in the drift, work in the stope is carried ahead from the last chute raise, thus determining the direction in which the drift should be driven. In the wide portions of the orebody, a second, and sometimes a third, drift is necessary to draw the ore evenly from the stopes. In other words, the main idea, after the ore is located

on a level, is to follow it, as local swells and pinches in the orebody and the method of mining followed preclude any definite layout of the haulageways as in lower grade and more regular orebodies.

In order to mine the ore on the extreme west end of the orebody, it is necessary to drive raises through the underlying dull gray and crinkley

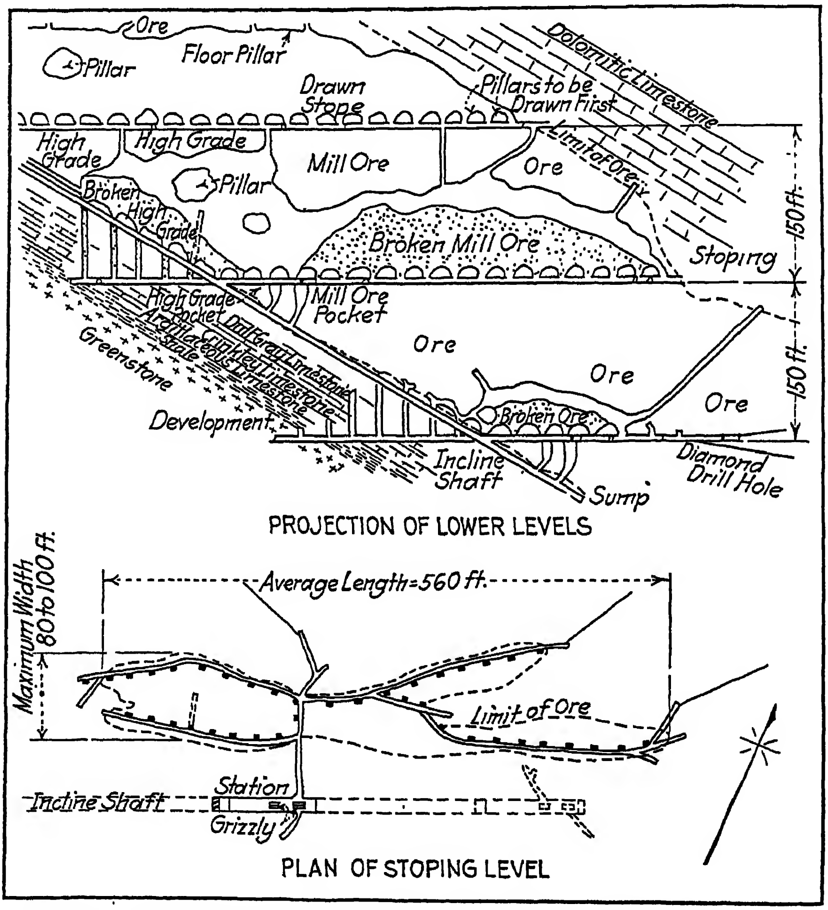

FIG. 2.—PROJECTION AND PLAN SHOWING GENERAL MINING METHODS.

limestone and the greenstone; when the levels are driven 200 ft. apart, a sublevel is driven to eliminate the long raises that would be necessary.

Fig. 2 shows, in plan and projection, a typical orebody and the development work required to stope it. The main haulageways are driven 7 ft. wide by 7 ft. high on a grade of 0.5 per cent. in favor of the loads; the prospecting drifts and crosscuts are 5 by 7 ft.; 16 and 30-lb. rails are used, the gage of track is 18 in. A compressor plant at Bonanza mine furnishes air for all the connected mines, a 6-in. line being used.

Very little timber is used, only an occasional set being necessary in passing through faults or on the greenstone contact; usually native round timber is used with round poles for lagging.

Loading machines are used in driving the larger headings; while they expedite the removal of the broken material, thus avoiding any delay when the miners are ready to set up for the lifters, a crossbar being used, they have not reduced the cost per ton removed. Scrapers are used at the Glacier mine, as noted; they are also employed advantageously when the main inclines are raised out, instead of being sunk.

Tramming is done by hand, horse, and storage-battery locomotives. Hand tramming is used where the distance is short and a small tonnage is moved; horse tramming, when the distance is greater; for the long hauls and on the levels producing the greatest tonnage, 4-ton Baldwin-Westinghouse locomotives with Edison cells are used. This type of locomotive has given very satisfactory service.

For horse and hand tramming, 20-cu. ft. end-dump cars are used; with locomotives, cradle-type and side-dump cars of 36 cu. ft. capacity are used, usually in trains of six or eight cars. While, on several levels, the locomotives run on 16-lb. rails, the practice is to use 30-lb. rails; curves have a minimum radius of 40 ft.

Hoisting is done in balance, the hoists at the Jumbo and the Bonanza are duplicates; they are of single-reduction, herringbone-gear type with a rope speed of 600 ft. per min., driven by two 85-hp., a.c., 2200-volt, three-phase, sixty-cycle motors; they were manufactured by the Allis Chalmers Co. The Mother Lode incline will be equipped with a double-drum hoist, with double reduction gears, driven by two 75-hp. motors; the rope speed will be 450 ft. per min. The cables are six-strand, nineteen-wire, Lang lay, ⅞ in. in diameter. When hoisting men, the skips are removed and a man car used. Neither skip nor man car is fitted with a safety device, as a satisfactory one has not yet come to the company's attention.

The air-compressor plant furnishes air for all mines, except the Erie, where an Ingersoll-Rand Imperial type 10, 600-cu. ft. capacity, electrically driven compressor is installed. The plant contains: One Ingersoll-Rand type P. E.–2 compressor, 1500 cu. ft. capacity, driven by a 250-hp. synchronous motor; two Ingersoll Rand Imperial type 10 compressor, 500 cu. ft. capacity, each driven by a 85-hp. motor; one Ingersoll-Rand Imperial type 10 compressor, 650 cu. ft. capacity, driven by a 105-hp. motor.

Because of the numerous openings to the surface, natural ventilation, with the exception of small fans belt-driven by a 10-hp. motor in development, aided by doors to course the air, is satisfactory.

Electric lights are used on the levels and incline shafts. The miners use carbide lamps, furnishing their own caps and lamps, the company keeping them in repair.

Each level has a telephone connecting with the foreman's office, compressor room, and hoist room. The mine telephone system is independent of the general system.

Electric pull bells, modeled after those commonly used in other mines, are used.

Operating Data

Types of Drills

For drifting Ingersoll-Rand, 248 Leyner machines are used; for stoping and raising, Ingersoll-Rand C. C. 11, except when drilling in chalcocite, when it is necessary to use a water-type drill. Ingersoll-Rand B. C. R. 430 and Sullivan D. P. 33 are used for blockholing and where occasional flat or down holes are to be drilled. Four-point, cross, high-center drill bits are used on all machines, made up of the following sizes of steel: 1-in. quarter octagon for stoper; 7/8-in. hollow hexagon for Jackhamer; 1 1/4-in. hollow round for Leyner. The bits are:

Stoper, 1 7/8-in. for starters; 1 3/4-in. for seconds; 1 5/8-in. for thirds; and 1 1/2-in. for fourths.

Leyner, 2-in. for starters; 1 7/8-in. for seconds; 1 3/4-in. for thirds; 1 5/8-in. for fourths.

Record of Unit Production

(*a*) Ore broken	297,502 short tons
Ore produced	294,202 short tons

Labor Data

(*c*) Stoping labor includes: Miners in stopes, muckers in stopes, bulldozers in stopes, rockbreakers in stopes:

Tons broken per man per hour	1.3964
Man-hours per ton	0.7161

(*d*) and (*e*) Exploration and development labor, miners only:

Tons broken per man per hour	1.2944
Man-hours per ton	0.7726

(*g*) All underground labor including above labor:

Tons produced per man per hour	0.4586
Man-hours per ton	2.1807

(*h*) Surface labor, exclusive of office force:

Tons produced per man per hour	11.5536
Man-hours per ton	0.0866

(*i*) All labor including office force:

Tons produced per man per hour	0.4243
Man-hours per ton	2.3570

	Per Cent. of Total
(*j*) Classification of Labor:	
Stoping, as in (*c*)	30 72
Exploration and development, as in (*d*) and (*e*)	6.04
Other underground	55.76
Total underground	92.52
Surface, miscellaneous	3 67
Office	3.81
Total surface	7.48
Total mine	100 00

(*k*) Labor turnover for mines.................... 161 per cent.

(*l*) Labor cost, 55.2 per cent. of total mining cost.

Supplies Data

(*a*) Explosives, 1.4713 lb. per ton of ore produced.

	Horsepower-hours per Ton
(*c*) Mining	4.974
Haulage and hoisting	4.961
Pumping	0.112
Ventilation	0.181
Lighting	1.273
Total	11.501

(*d*) Cost of supplies (exclusive of explosives and power), 22.6 per cent. of total production cost.

(*e*) Cost of all supplies, 44.8 per cent. of total production cost.

Safety Measures

Hoisting ropes are thoroughly inspected once each week; every six weeks 2 ft. are cut off from both ends. The ropes are changed end for end after six months' use. The sheaves are inspected once every week, and the hoists each day. In the vertical shaft, the safety catches are tested every Sunday.

There is a fire extinguisher on every level station; fire doors are provided. When located near timber or snow sheds they are of concrete and steel; otherwise they are built of wood, care being taken to make them as air-tight as possible.

No safety engineer is employed, the engineering department reporting to the general mine foreman and superintendent any unsafe practices that come to its notice.

Each bunk house is equipped with a pool and reading room, a number of magazines and other periodicals being provided. Moving picture shows are given twice a week at Jumbo and Bonanza camps.

A well-equipped hospital is located at the mill camp with a competent surgeon and corps of nurses in attendance.

At the plant, last year, there was one fatal accident; no serious accidents causing total permanent disability; three partial permanent disability; 28 causing loss of more than 14 days' time; and 120 minor, loss from 0 to 14 days.

Compensation paid, under Territorial Act, amounted to 1.058 per cent. of the payroll.

DISCUSSION

R. M. Raymond,* New York, N. Y.—Is any estimate made of the orebodies in the reports or for the information of the stockholders or the officers? If so, on what basis is it made?

Alan M. Bateman,† New Haven, Conn.—Estimation of ore reserves is a difficult matter, when dealing with orebodies of such irregular shape and containing unusually high copper content minerals. The company formerly kept assay maps on which were plotted results of samples of every 5 or 10 ft.; but with such rich ore of this type only a slight irregularity of the vein will greatly change the sample, so it was found that such assay maps were not dependable and not worth the time and expense they required. Plain, straight eye estimation of the grade of the stope on the part of a good superintendent was found to be as reliable as estimates obtained from assay maps. A more careful check than this, however, has always been made. When running a drift in ore, each car would be sampled; also when drawing a stope, the cars would be sampled. More recently, the ore when being dumped from the skip passes over an automatic sampler, in which a large scale sample is obtained.

When estimating the ore reserves, care must be used in dividing the ore according to the different grades. Mill ore will contain from 5 to 15 per cent. copper, and the high grade may contain from 30 to 60 per cent. copper directly as mined. The estimates are usually made extremely conservative, with a guarantee of safety to the operators and officers of the company.

R. M. Raymond.—As to the sampling of any of the cheaper or milling ores and the distance one could reach out into that, was there any attempt to explore by drilling or otherwise in order to learn the extent or depth of the orebody?

Alan M. Bateman.—The company has done about 80,000 ft. of diamond drilling in these mines, the chief purpose of which is to prevent missing any ore laterally. Drilling in depth has been carried on in

* Professor of Mining Engineering, School of Mines, Columbia University.

† Professor of Economic Geology, Yale University.

advance of shaft sinking, to determine the location of the orebodies on the lower level. Such drilling, however, has usually been to outline in depth only the ore immediately below the lowest level or so. Short diamond drill holes are run out from the drifts and stopes. These holes test the walls of the stopes and drifts to make sure, before any stope is drawn, that no orebody is overlooked. Most of the exploration is by means of the direct crosscuts, as the geologic conditions that control these orebodies are fairly well known, and exploration can be directed accordingly.

Enoch Perkins, Wharton, N. J.—Will Mr. Bateman tell us of the long transfer tunnel from one mine to the other and if the storage batteries used are satisfactory?

Alan M. Bateman.—The tunnel was run primarily as an exploration crosscut tunnel; later it was used to transfer ore from the Bonanza to the Jumbo mine. When operating control of the Mother Lode mine was obtained by the Kennecott company, these mines were also connected by the long crosscut, and all the ore was transferred by means of storage-battery locomotives. In the latter crosscut, these storage-battery locomotives are run daily, and as far as I am aware, with entire satisfaction.

Enoch Perkins.—Anybody who has run storage-battery locomotives can appreciate the difficulties we formerly had with them. They must have a type of storage battery that permits tramming under extremely cold temperatures.

Alan M. Bateman.—That is an erroneous conclusion. While the outside temperatures, in winter, are extremely low, the mine temperature is constant, and is just about at the freezing point. This is a rather unusual condition, for to a depth of 600 or 700 ft. the ground water is frozen and the cracks are filled by ice. No water exists. The temperature is about 30–32° F., which incidentally is the best shift boss that has ever been found.

Mining Methods of the Cripple Creek District

BY FRED JONES, VICTOR, COLO.

(New York Meeting, February, 1923)

THE Cripple Creek district is in Teller County, Colo., about 18 miles in an air line west of Colorado Springs and at an elevation of 10,000 ft. A line drawn through Colorado Springs and Pikes Peak will, if continued, about pass through the district. The character of the country generally is rolling, with few trees and little snow fall. The annual precipitation is about 20 in. All the water used by the mines is supplied by the Altman Water Co., which pumps from the west slope of Pikes Peak.

Gold was discovered in 1891, since which time the district has produced $411,000,000. The peak of production was reached in 1900 and amounted to $25,000,000. At present the production amounts to about $400,000 per month.

GEOLOGY

In general, the geology of the district is simple; ash veins and dikes traverse a volcanic plug of breccia. The ore occurs as enriched spots along these dikes and veins. The dikes represent the most recent extrusions and consist of basalt and phonolite. (For a complete geological description reference is made to Professional Paper 54, 1906, of the U. S. Geological Survey.)

The ore consists of small parallel seams of high grade. In only a few cases is the gold disseminated through the rock; much of the gold in the broken ore is found in the fines. The wall rock is hard blocky granite or breccia. About 25 per cent. of the rock mined is shipped as ore to the Golden Cycle mill at Colorado Springs.

The height of an ore deposit is usually several times its length. Few flat oreshoots have been found. The veins are all steep and have a dip of 70 to 80°.

DEVELOPMENT

The claims are all 300 by 1500 ft. and are held by location or patent. All mining is done through shafts, one of which has reached a depth of 2600 ft. The main working shafts are 4½ by 13⅛ ft. and are made up of three compartments, each 4 by 4½ ft. in the clear. They are timbered with 10-in. Oregon posts and plates and 8-in. spreaders. Two compart-

ments are used for hoisting in counterbalance, while the third is reserved for dinkey way, power lines, and air lines.

The levels vary from 75 to 150 ft. apart. Drifts are driven 5 by 7 ft. with a grade of 0.5 per cent. in favor of the load; 12-lb. rails, and 1¼ to 3-in. air lines are used.

All drifting is done on the veins. In the Portland, the lower levels average 35 tons of ore per foot of development; this does not include what will be gleaned by lessees a little later.

Exploration, Sampling, and Estimating

When opening new levels and exploring for new ore, a crosscut is run through the country to find the veins. They are then drifted on to open the oreshoots. Some diamond drilling has been done, mostly to determine geologic conditions. The orebodies are so spotted that diamond drilling is not considered the best method of exploring new ground.

Samples are taken by shift bosses. The streaks are usually sampled together with a couple of grabs from the muck pile. Ore drawn from chutes is sampled by taking a handful from each car trammed. These grab samples will run about 20 per cent. higher in value than the settlement value of the ore. The estimates of the value of ore shipments, made by the ore-house foreman, are fairly close to the realized value—always within, say, $3 per ton.

When estimating the value of a block of ore, cuts are taken across the streaks, with an occasional sample of the adjacent rock. The ore in sight is measured in the Portland every 3 months and the tonnage mined invariably runs more than the estimate, because the stopes break wider than expected and the grade is correspondingly reduced. The tonnage mined runs about 20 per cent. higher than that estimated.

Only a few of the properties have resident surveyors. This work is done on contract by deputy mineral surveyors. The Portland plan maps are on a 60-ft. scale and the longitudinal sections on a 50-ft. scale.

Tonnages are calculated on a basis of 13 cu. ft. to the ton for ore in place and 22 cu. ft. per ton for broken ore.

Mining Methods

In the early days, the wide orebodies were mined by square sets, but to save timber this method has been replaced by the shrinkage system. The narrow stopes are also worked by the shrinkage system in preference to the old stull system, because of greater safety and economy. Ninety-nine per cent. of the stopes will run from 3 to 10 ft. in width; the length sometimes reaches 1000 ft. In the cut-out the back is carried about 15 ft. high. The level stulls are then put in 7 ft. in the clear with chutes 18 ft. apart. The stopes are then carried through to the next level on broken ore. About 40 per cent. of the ore will be drawn by the time machine

work is finished. Timbermen then follow the stope down as the ore is drawn and catch up loose ground and clean down the walls; 6 to 8-in. stulls are used for this purpose.

Drilling and Blasting

Ingersoll-Rand No. 248 or some style of Waugh water machines are used in drifting and crosscutting. Round hollow steel 1⅛ or 1¼ in. in diameter with an ordinary cross bit is used. Electric blasting with battery or lighting circuit is used in shafts and wet places.

Ingersoll-Rand and Waugh stopers (hammer drills) are about the only types of machines used in the stopes. The Portland company has adopted the wet machines because they drill more holes, break less steel, and keep down the dust. The 1-in. quarter-octogon steel is used. The men prefer the wet machines after they have used them a while.

In shafts and tough drifts 40 per cent. gelatin dynamite is used and in stopes, 30 per cent. dynamite.

An average of 1.18 lb. of explosive is consumed per ton of ore hoisted.

Timbering

Little timbering is needed in drifts and crosscuts. Shaft timbers are treated with creosote and no timbers are recovered for re-use. Little timber is used in stopes and few of them are filled. The waste from development and shaft sinking is put into old stopes but falls far short of filling all of them.

Haulage

Storage-battery and trolley locomotives are used for underground haulage. The storage-battery motors are 3-ton and the trolley motors are 8-ton size. Cars of 1500-lb. capacity are used with the former and 4-ton cars, Granby type, are used with the latter. The rails vary from 20 to 60 lb. per yard.

Hoisting and Air Compression

Three and four-ton skips are used with loading pockets in some of the mines. Some mines have double-deck cages with single cars on each deck. Loading pockets are in general use and vary in size from a few cars to nearly 1000-car capacity.

The large hoists are of the flat-rope, Wellman-Seaver-Morgan steam type, ranging in size from 10 by 12 to 24 by 48. The rope speed is about 3000 ft. per min. The ropes are ⅜ in. thick and 5 in. wide. The largest hoist in the district is at the Portland No. 2 shaft; it has a rope capacity of 3500 ft. Almost all of the steam compressors have been replaced by direct-connected high-speed electric machines of about 1100-cu. ft. capacity.

Drainage

The Roosevelt deep drainage tunnel cuts the mines at a maximum depth of 2200 ft. and drains the entire district. It is about 6 miles long and is used solely for drainage. There are only two mines in the district operating below this level. The Portland, 500 ft. below, handles 1000 gal. of water per minute. The increased depth has added little to the flow. This company uses one three-stage, Buffalo, centrifugal pump driven by a 200-hp., 2200-volt, General Electric motor. Two similar pumps are held in reserve.

Ventilation and Lighting

Natural ventilation is depended on almost entirely. Some of the small mines work under pressure of a few ounces to keep out the CO_2 during periods of low barometric pressure. (A complete description of the method has been given by S. A. Worcester in "Pressure Ventilating System used at Cripple Creek," *Eng. & Min. Jnl.*, June 8, 1915, p. 981.) The upper levels are drafty and cool because of the many connections from one property to another. The 2600-ft. level of the Portland is considered warm with a temperature of about 75°.

All of the large mines are lighted with electricity and have mine-a-phones connecting the levels with the different departments on the surface. Miners carry carbide lamps.

Unit Production and Costs

The records of the unit production are taken entirely from the operations of the Portland mine for August and September, 1922. The total production for this period was 29,674 short tons. It required 57,056 man-hours to produce this quantity, or 0.52 ton per man-hour. This includes all men on the payroll, even to the manager. As 33,600 man-hours were worked underground this gives 0.88 ton per man-hour for underground labor; 23,456 man-hours were worked on the surface or 1.26 tons per man-hour on the surface.

The production per man-hour is 2.4 tons for machine men in stopes and 2.25 tons for trammers, including the time in mucking cut-outs.

The percentages of different costs are as follows:

	Per Cent.
Payroll (labor)	61
Store house	8
Timber	1
Fuel	11
Explosives	11
Power	7
Miscellaneous	1

The mining cost is 64 per cent. of the total cost of mining, freight, and treatment.

Material is hauled by the Midland Terminal R.R. from Colorado Springs, a distance of 45 to 50 mils. The freight rate on ore to the Golden Cycle mill is $1 per ton for a grade of $200 or under. The rate on coal averages about $3 per ton.

All mine timber is shipped from Oregon or timbered districts of Colorado at a cost of $35 to $40 per 1000 ft.

Labor

The district is known as the "white man's camp" and has efficient labor. The nationalities represented are Americans, Swedes, Cornish, Welsh, and Irish. There are practically no "dark-skinned" laborers.

Every man is required to have a Mine Owners Association card before he receives employment. There are no unions. The percentages of different classes of labor are as follows:

	Per Cent.
Underground	54
Mechanical	30
Ore house	8
Office	8

The labor turnover will amount to about 0.33 per cent. monthly or, say, 4 per cent. per annum. The scale of wages is as follows:

	Per Shift
Machine men	$4.25
Timber men	4.25
Cagers	4.25
Hoistmen	5.00
Firemen	4.25
Trammers	3.75
Compressor men	4.00
Top men	3.50

Fifty cents is added for work in wet places. Few men are working for this scale as all the work is contracted—stopes by the fathom, drifts by the foot, and tramming by the car. Men on contract average from $5 to $12 per day.

DISCUSSION[1]

Fred Jones.—Different scales are used on the plan maps and the sections because this had become the custom. When necessary to use both plans and sections, the glass model was used instead of maps. The only men who object to the creosote are the shaft men; there are no complaints after the timbers are in place. The figures for tons per man-hour refer to crude ore. The figures for power refer to electric power

[1] Offered at meeting of Colorado Local Section.

only; fuel refers to steam power. In contract work, the company furnishes all tools and supplies except explosives.

Lessees usually made little more than wages while contractors made from $5 to $12 per day and averaged $6.81; lessees, however, always have the hope and possibility of making much more.

The longest haul for which storage-battery locomotives are used at the Portland is 3500 ft. They have been used for about 11 years and are highly satisfactory for long hauls. In hauling from the Lost Anna stope, two men hauled 125 cars of 16-cu. ft. capacity per shift a distance of 2500 ft. and dumped the cars in the shaft pocket. They were paid 7 or 8 cents per car. The upkeep cost for these locomotives per ton of ore is almost too small to calculate. The batteries are recharged every day, and where the haul is long the battery is connected with the charging current while the men are at lunch. Edison batteries are used.

F. G. Farish.—At a mine in California, they have successfully used Port Orford cedar instead of creosoted pine.

Fred Jones.—The present production of the Cripple Creek district is about $400,000 per month. The Cresson mine is flourishing. The grade of the ore is fairly good in general. High-grade streaks in the bottom level of the Portland run as high as $60 per lb. The entire oreshoot on this level is about 1000 ft. long and has been continuous from the surface. The force had been gradually reduced from 300 to 150 men; at present 1200 or 1500 men are working in the mines. The plan of having all blasting done by a firing squad, which was tried during the war, resulted in the miners and the shotfirers blaming one another for poor results. The most satisfactory method is to have one set of men responsible for the whole work in a given place.

F. G. Farish.—At one mine it has been the custom to whitewash the face immediately after the shots. Hand-rotated wet stopers were used as the self-rotaters were too heavy and too complicated for satisfactory use.

G. E. Collins.—Stopes could be carried narrower by using self-rotaters.

Fred Jones.—By using care with the water feed, the men need not get wet above the waist; they wear waterproof overalls and rubber shoes.

Mining Methods of Jarbidge District

By John Furness Park, Jarbidge, Nev.

(New York Meeting, February, 1925)

The mining district is located in the northeasterly part of Nevada, between the Jarbidge River on the west and the East Fork of the Jarbidge on the east. The northern boundary of the district is but a few miles south of the Idaho state line—the gold-bearing area extending from this point south, for 5 or 6 miles.

The Jarbidge Mountains, in which the ore deposits occur, range from 5000 to 11,000 ft. in elevation, the most important mines being situated about 7000 ft. or higher.

The town of Jarbidge lies along the Jarbidge River and is reached by stage from Rogerson, Ida., the nearest railroad point. During the summer months, the camp may be entered from the south by way of Deeth, Nev., but no regular transportation facilities are available over this route.

History

The mineral deposits were not discovered until 1909, when D. A. Bourne, panning along the river bank, found gold in the gravel at the mouth of a small stream flowing into the Jarbidge. He prospected in the vicinity of this stream, later known as Bourne Gulch, and shortly after located the North Star claims.

During the following spring and summer (1910), the camp of Jarbidge grew from nothing to a town of 2000 people and the hills surrounding it were practically all taken up by mining claims.

Most of the work done during the first year or two consisted of trenching and open-cut prospecting. Tunneling was started on a number of claims and many veins were uncovered. Mills were built and operated on a few properties, notably the Pavlak and the North Star. These ran only a few months, however, when they were closed down because of extraction and financial difficulties.

A period of development followed in which the majority of the present producing mines were opened. The Elkoro Mines Co. acquired a number of claims in 1915 and started work on the present mill in 1917. Since that time other properties have been taken over until this company is by far the largest operator in the district. Only one other company is now actively engaged in mining, the Bluster Mining & Milling Co., which has

but recently completed its mill. As mining operations have been under way but a short time, few data are available from this source. Several other companies are opening new territory but none has reached the producing stage.

Unit Size of Mineral Tracts

The gold-bearing area of the Jarbidge district does not exceed 14 sq. mi. This section is, at present, divided roughly into twenty-five or thirty main claim groups. These groups include about forty veins, of which perhaps half are extensively developed, and which vary greatly both in length and width as well as in value. Most of the veins found have not proved of commercial value because of their location or grade of ore.

The vein systems of the district may be classed in two sections—those dipping to the east and those dipping to the west. The former have been the more productive and are of greater width, running as high as 30 ft. while the latter seldom exceed 4 ft. between walls.

The ore-bearing zone of the veins so far developed does not exceed 700 ft. vertically, although the veins themselves extend to a much greater depth.

In length, the veins vary from a few hundred feet to several miles. They are cut by numerous faults throughout their length and are never of continuous commercial value.

The claims are held by patent and are 600 by 1500 ft. Several groups of claims are leased under special mining and milling agreements with the owners.

Geology of District

The ore occurs in a series of Tertiary volcanic flows that have undergone extensive weathering and alteration. The lavas have been divided into two main sections, known as the older and the younger rhyolites. Each of these is in itself a series of flows of varying thickness and importance. They are again divided into three classes—upper, middle, and lower. Of these, the most favorable horizon has been found to be the upper section of the middle rhyolite and the lower section of the upper. Although small bodies of ore have been found outside of the so-called favorable horizon, they have never been large enough to prove profitable. This section does not often exceed 650 ft. in thickness. An extensive and detailed description of the flows of the district has already been given.[1]

The gold and silver are found, mainly in the native state, in quartz veins that are generally well defined and are nearly vertical, having an average dip of from 70° to 80°. Although these veins show great variance in size and value, the same main characteristics seem to hold for all of the

[1] Frank Charles Schrader: A Reconnaissance of the Jarbidge, Contact, and Elko Mining District, Elko County, Nev. U. S. Geol. Surv. *Bull.* 497; The Jarbidge Mining District, Nevada. U. S. Geol. Surv. *Bull.* 741.

forty or more discovered in the district. The chief gangue minerals are quartz and adularia of which the former is by far the more important. The veins are generally soft and often contain streaks of clay or gouge, up to 6 or 8 in. thick. Near the surface, the gangue minerals are usually extensively stained by iron and manganese oxidation products.

Practically all the mining done up to the present time has been carried on in the oxidized section of the veins. One company is now doing development work in the sulfide zone, where a small quantity of ore has been discovered. It is hoped that further work in this zone will uncover larger orebodies although the experience gained in other mines would seem to prove this unlikely.

Items That Influence Mining Methods

The mines of Jarbidge occur in such a steep rugged country that all the veins can be reached by horizontal tunnels driven from the surface. These tunnels are placed at about 100-ft. intervals and serve as both working and development bases.

Because of the heavy snow fall in the winter, sufficient timber and lumber is brought in during the summer months and held in readiness at the mines. The timber used is mainly a native pine which, although of no great strength, serves the purpose as no great weights have been encountered and the veins are not large enough to open large areas when the ore is removed.

All mining material and machinery must be freighted from the nearest railway point, a distance of 70 miles. This is done at present by motor trucks at a cost of 1¾ cents a lb. The cost is added to by the necessity of hoisting the material to the mines, which are situated from 800 to 1700 ft. above the town. This work is done by surface tramways, using cars during the summer and sleds when the snow becomes deep.

The miners employed at Jarbidge are, in the main, native born and are not unionized.

Exploration, Sampling, and Estimating Methods

Exploration is now carried on mainly by means of drifts and crosscuts. Diamond drilling has been used, but the great number of slips, faults, and places of low value occurring in the rock made the results unreliable. The hole in some cases may pass through a fault zone and the core show values where no vein material exists. Most of the rock encountered in such work is too soft or broken up to make a good core so that very little can be determined by the appearance.

Because of the rather thick layer of debris covering the surface, trenching has not proved successful although several veins have been uncovered by this method.

Prospect tunnels are driven in the favorable horizon and the veins followed and carefully sampled. The wall rock is investigated by means of crosscuts and by holes drilled by ordinary drifting machines using extension bits. These holes may be quickly and cheaply drilled to depths of 50 or 60 ft. and the sludge samples obtained show the presence of any values equally as well as the more expensive diamond-drill hole.

Drifts, driven on the vein, are sampled at intervals of from 3 to 5 ft. The samples are taken with a pick and average about 10 or 12 lb. in weight. The face is sampled in sections, usually three or more samples being taken in one cut across the width of the drift—one sample representing the main vein material and the others stringers or wall rock included in the opening. A further sample is taken of the muck removed from each round; this serves as a rough check on the samples cut from the face.

The assay values of the samples are averaged to show the average value of the drift for its total width. These values are plotted on a plan map to the scale of 1 in. to 10 ft., which map is kept up to date as the work progresses. The same method is used with raises and winzes, and, when making estimates, any unusually high values are discarded.

Raises are put up in ore at points along the drifts determined by working conditions, usually about 100 ft. apart. The blocks between these raises are estimated from the average assay values and the cubical contents. The tonnage figure for the ore is 18 cu. ft. per ton in place and 23 cu. ft. when broken. The estimate of the tonnage in a block is generally within 10 per cent. of the actual amount obtained and is always low, because of the dilution in mining the ore. The estimated value runs about 17 per cent. higher than the true value of the ore obtained, which factor is taken into account when making reports on newly blocked-out territory.

In addition to the assay maps, others on a scale of 20 ft. to the inch are used showing the geology and working features of the tunnels. Also maps on a scale of 100 or 200 ft. to the inch, on which a large part of the workings are shown on a fairly small sheet, are used in and around the mines to show the relation between the various working places. A glass model in the main office shows all the work done up to the present time, together with the faults and veins in their true positions.

Underground Mines

General Features

The veins are usually narrow, being from about 3 to 10 ft. in width with occasional short stretches running up to 20 and 25 ft. The walls of rhyolite are fairly strong and generally merely require stulling, when the ore is drawn from the stope, to steady loose sections. The average dip

is between 75° and 80°, so that little trouble is experienced in drawing the ore.

The orebodies vary greatly in length and height. The longest minable section so far encountered is 700 ft., while the vertical component of the commercial ore has run between heights of 50 and 450 ft. The veins are of greater extent than this but barren sections and faulting cause portions of them to be unprofitable.

The ore runs from soft to medium hard; often both are found in the same stope and within short distances of each other. The values are best called spotted, although a series of samples taken from a back forms a good working basis, as the spots of high and low value, although occurring in different places as the back is raised, are usually compensating when the general average is figured.

Occasional horses of waste are found within the orebodies but are seldom large enough in volume to make sorting necessary in the stopes. When these sections are large enough to cause serious dilution of the ore, they are left in the form of pillars.

No high temperatures or dangerous gases are encountered in the Jarbidge district and natural ventilation is sufficient, except in long development drifts and winzes, where fans of both the pressure and the exhaust type are installed.

Mine Openings

Practically all mining is done from drifts driven from the surface. Considerable development work has been done from winzes but at present

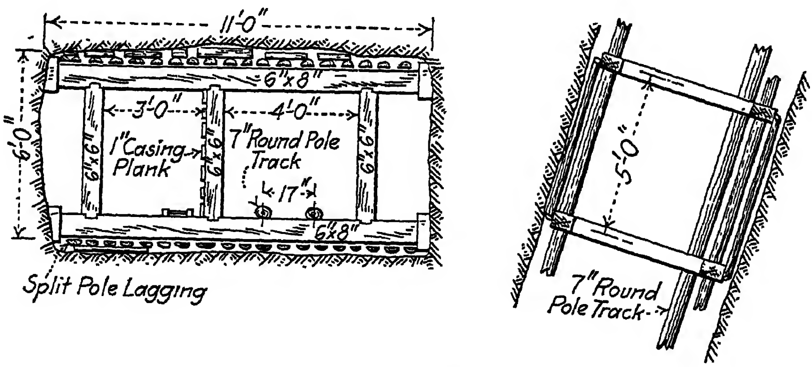

Fig. 1.—Method of timbering winzes.

all the ore taken from the mines is obtained in stopes above the adit levels and none is hoisted.

The main drifts are generally 5 by 7 ft. in the clear and, where necessary, are timbered with round timbers of native pine. Development tunnels in waste are driven only large enough to accommodate the cars and workmen; they are usually 4½ by 6 ft.

The winzes, from which waste from development work is hoisted, are of two compartments; the skipway is 4 by 4 ft. in the clear and the manway 3 by 4 ft. They are timbered with squared native timber, as shown in Fig. 1.

The bucket track is made of 7-in. round poles framed to fit snugly at the joints; these give good service and are easily replaced.

Rock is hoisted in buckets of 700 and 900 lb. capacity, which are loaded from a pocket or placed on trucks and run to the working face, where they are filled by hand. Automatic trips are placed at the top of the winzes and the buckets dump into small bins, from which the muck is trammed to the dumps outside the mines.

As it is only necessary for the pockets to hold the muck that is taken from the face of the tunnel, they are generally of from 20 to 30 tons capacity.

Underground Development Plans

Main levels, opening on the surface, and sublevels are driven in ore at intervals of 100 or 150 ft. Raises are put up at points where working conditions make them most favorable or where the sampling shows that further prospecting is advisable. The distance between these raises varies from 100 to 200 ft. They are carried the full width of the vein and are later used as manways or ore passes during stoping operations.

All track used is of 18-in. gage; 12-lb. rails are used on main levels and 8-lb. in development tunnels. A grade of 0.5 per cent. in favor of the load is maintained.

Water is run, on the levels, in ditches at the side of the track and piped over openings. No great quantity of water has been encountered and special facilities for conveyance are unnecessary.

Compressed air is carried into the mines in 4- and 3-in. pipe; 3-in. is used on the main levels with 1½- and 1¼-in. air lines running up raises and into stopes. For long development drifts, 1¼-in. and occasionally 1-in. lines are used.

Where the natural ventilation is not sufficient, 6- and 8-in. galvanized ventilating pipe is used.

Mining

In drifting and crosscutting, Denver rock drill, mounted water machines (Dreadnaughts) are used, except where the ground is fairly soft when jackhammers of both the Waugh and Ingersoll-Rand makes are mounted. The smaller hole drilled by the jackhammer makes it necessary to put in several more holes to break a clean face; but this is offset by the lightness of the machine and the fact that when a cross bar is used only one set up need be made as the lifters are drilled from a plank laid on the bottom after the last muck is removed. In most of the drifting,

drilling is done on two shifts so that an upright bar cannot be used as too much time is taken up in mucking back for the set up.

Practically all stoping is done with the Ingersoll-Rand CC-11 dry stopers, although the Waugh hand-rotating wet machine is used where dust is an important feature.

The following types of steel are used with the various machines:

Large drifting machines..........	1⅛-in. round hollow steel.
Jackhammer drills.......	⅞-in. hexagon steel.
Stopers..........................	1⅛-in. cruciform.

Steel is sharpened with a double-taper cross bit, using a ⅛ in. change in gage with each change in length.

No standard round is used in drilling as the conditions of the rock vary too much in short distances. In drifting, the ordinary down cut is generally preferred by the miners. In the stopes, a V cut is placed in such a position that the force of the shots, as far as possible, will be away from manway timbers or pipe lines.

At present, 40 per cent. gelatin is used in all blasting; 1-in. powder for jackhammer holes and 1⅛-in. for all other work.

The shrinkage method of stoping is used entirely in the Jarbidge district. Drifts are first run in the vein, taking the full width of the ore. These are back stoped, stulled, and lagged. Chutes are built every 15 ft. or as close to this interval as local conditions will allow. Manways are placed about every 100 ft. This is not a definite figure, however, as manways put up in development raises are used whenever available.

The stopes are carried up as high as 250 ft. with very little timber, except that used in the manways and for staging.

About 23 per cent. of the ore is drawn from the stope during stoping operations. When all the rock has been broken, the ore remaining is drawn out and the walls cleaned and stulled by timber crews working on top of the muck as it descends.

No sorting is done in the mine. The ore is dumped upon sloping grizzlies covering the outside ore bins and large pieces of waste rock are thrown out from a breaking platform at the lower side. The ore passes through other grizzlies in the mine, but they serve merely to prevent large boulders from clogging the chutes.

Timbering

Where support is necessary in drifts, stopes, and raises, round timbers are used. Shaft sets are framed from 6 by 8 in. and 6 by 6 in. native pine. The lagging used in manways is generally 3-in. plank, but occasionally split poles are used. Stull timber and drift sets are cut outside the mine and delivered to the working places by trammers as needed. Small

sticks used to support loose slabs when timbering down stopes are usually sent in in pole lengths and are cut by the timber crews in the stopes.

Timbers are hoisted in the manways by air hoists of the "tugger" type. Small skips operating in timber slides of 2-in. plank carry the material necessary for the stopes.

No preservatives are used and practically no timber is recovered.

Underground Sampling

Sampling is done in the mines by one man, who visits each working place and samples the fresh face or back after each round. The samples are taken by hand at 5-ft. intervals or closer. In drifts, samples are taken after every round; and in crosscuts, a cut is taken continuously along the side. In addition, a grab sample is taken from each car of muck by the trammers or muckers, and the samples thus obtained are collected and recorded by the sampler.

A final car sample is taken from the train delivering ore to the outside ore bins, which serves as a check on the mill heads.

Tramming and Haulage

On all the main adit levels of the Elkoro Mines Co., rock is removed in side-dump, Sanford Day, cars of 26 cu. ft. capacity. These are hauled in trains of three to ten cars by mules or electric locomotives; the G. E. storage-type motor is used for this work.

On other levels, cars of 16, 18, and 20 cu. ft. capacity are used in handling ore. These cars are of the end-dump type and are run both in trains, with mule haulage, and by hand. On all levels where ore is trammed, 12-lb. rails are used.

Storage and Dumping

Ore is dumped into raises running from the main haulage levels to those above. These raises serve as storage pockets and hold from 200 to 300 tons of broken ore. Grizzlies at the top prevent large rocks from entering and stopping the flow of ore in the raises. The ore is drawn from chutes at the bottom, directly into the train and carried to the outside bins.

Hoisting of waste from development drifts and crosscuts is done by means of small, single-drum electric hoists of from 15 to 25 hp., using ½- or ⅜-in. cable. The muck is hoisted in buckets at a speed of about 250 ft. per min. and dumped into pockets of from 20 to 40 tons capacity. As the longest lift is 400 ft., the rope speed mentioned has been sufficient.

Very little pumping has been done in the district. At present, but one pump is in use—a Cameron No. 3 air-driven sinker. This is at work in a development winze and handles the water in from 6 to 7 hr. All other water encountered is run from the tunnels in ditches.

Compressed air is supplied by Ingersoll-Rand "Imperial" type 10 compressors. These are belt-driven and are of various sizes. The air lines running into the mines are connected to one another so that the compressors will run under an even load when the drain at one mine is greater than at the others.

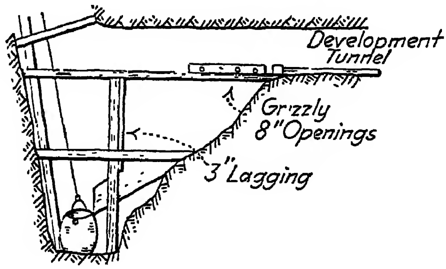

FIG. 2.—LOADING POCKET.

Tunnels, where mule haulage is used, are lighted by means of 110-volt circuits installed for the purpose. Hoisting stations are supplied with 220-volt lamps connected two by two in series and drawing their current off the 440-volt circuit used for running the hoist. All men underground use carbide lamps.

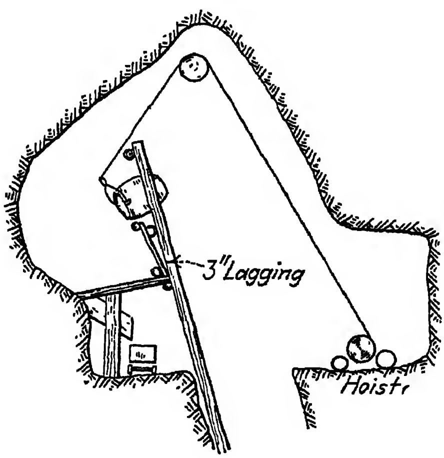

FIG. 3.—DUMPING POCKET.

UNIT PRODUCTION

The following records are given from the Long Hike mine of the Elkoro Mines Co. for the first six months of the year 1924. During this period, 21,550 tons (2000 lb.) of ore were shipped to the mill from the Long Hike and, in addition, 320 ft. of development raising, and 1244 ft. of development drifting and crosscutting were completed.

All stoping operations and tramming of ore was done on day's-pay, while the majority of the development work was done by contract. The contractors were paid by the foot advance, paying for their own powder.

TABLE 1

	Tons per Man per Hour	Man-hours per Ton
Miners in stopes (drillers only)	2.26	0.44
All men in stopes (drillers, timbermen, helpers, etc.)	1.04	0.96
All men in mine (including development workers)	0.418	2.394
Surface labor only (not including office)	4.34	0.23
All men (not including office)	0.406	2.466

NOTE.—As but one office is maintained for the several mines operated by this company, its unit cost cannot be accurately included in these figures.

TABLE 2.—*Development Work*

	Feet Advance per Man per Hour	Man-hours per Foot Advance
Drifting and crosscutting (all men)	0.149	6.688
Raising (all men, timbering included)	0.096	10.375

CLASSIFICATION OF LABOR

	Per Cent.
Miners (breaking rock)	18
Timbermen and helpers	18.70
Samplers	1.60
Outside labor (supplies, repairs, etc.)	3.40
Outside labor (blacksmiths, compressor men, etc.)	5.90
Trammers and muckers	17.80
Bosses	4.70
Contractors (raising)	9.50
Contractors (drifting and crosscutting)	20.40

During the period under consideration, the labor cost was 51.36 per cent. of the total cost of mining.

SUPPLIES USED PER TON OF ORE PRODUCED

Explosives (development work not included)	1.748 lb.
Explosives (development work included)	2.397 lb.
Timber (development work not included)	1.13 lin. ft.
Timber (development work included)	1.692 lin. ft.
Lumber (development work not included)	1.49 bd. ft.
Lumber (development work included)	2.10 bd. ft.

POWER

All power used in the Jarbidge district is supplied by the Idaho Power Co. and is brought to the camp over a high-tension line from the hydro-electric station at Thousand Springs, Ida.

TABLE 3.—*Power Used per Ton Ore Produced*

	Kw.-hr. per Ton
All mining	11.247
Compressed air	9.102
Haulage	0.126
Surface tramming (material and supplies for mine)	0.105
Lighting (includes lighting and heat of outside buildings)	0.913

The total cost of supplies was 30.26 per cent. of the total cost of mining.

Disposition of Ore

Ore is transferred from the bins at the mine portal to the mill by means of aerial trams. It is treated by a cyanide process and the bullion recovered is shipped by express.

DISCUSSION

R. S. Lewis, Salt Lake City, Utah.—The problem of minimizing the use of timber underground is recognized as an important problem in Utah mines. Many of the ore deposits are of such a nature as to require the use of square sets, for there is no other way to work them. Some mines use long caps, reaching across more than one set, and from wall to wall if possible. This serves to stiffen the sets and lessens the cost somewhat both for the framing and for the placing of the cap on the sets.

Mining Methods in Mogollon District, New Mexico

By S. J. Kidder,* Mogollon, N. M.

(New York Meeting, February, 1924)

The Mogollon mining district is in the southwestern part of New Mexico near the southern end of the Mogollon range, from which the district takes its name. The town of Mogollon is but 37½ miles in an air line northeast of Clifton, Ariz. The air-line distance from Silver City is 52½ miles and from Tyrone, 57 miles. Supplies are transported by motor truck from Silver City. The distance by road is 75 miles, half of which is an improved highway of modern construction. Transportation has always been a large item of expense in carrying on operations in the district.

Valuable ore is said to have been discovered in the Mogollon Mountains as early as 1870 but the first regular prospecting was carried on in 1875, by James C. Cooney, a sergeant from Fort Bayard. His discovery of rich silver-copper ores in the region of Mineral Creek led to the establishment of the Camp of Cooney. Remote from any settlements where there was a semblance of law and order, and harassed by the Apaches, development was slow and nearly 10 years elapsed before the village of Mogollon was established on Silver Creek. Prospecting in this section of the district had disclosed the outcrops of several promising veins containing silver-gold ores. Among the early claims located were the Last Chance, Top and Confidence, on what has proved to be the most productive vein in the district.

To the end of 1922, the Last Chance and Top claims had produced approximately 600,000 tons of ore and nearly 200,000 tons had come from the Confidence group of claims. The total value of the ore produced from this vein alone has amounted to about $9,600,000 in silver and gold.

The Fanney vein was developed later and during its most active period of production contributed more than half of the output of the district.

Other important locations, though far less productive than the above, were the Maude S, Deep Down, the Deadwood and the Pacific.

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* General Manager, Mogollon Mines Co.

All full-size claims in the district are 1500 ft. long by 600 ft. wide. The accompanying map, Fig. 1, shows the ownership of the principal claims and those under lease or option to purchase.

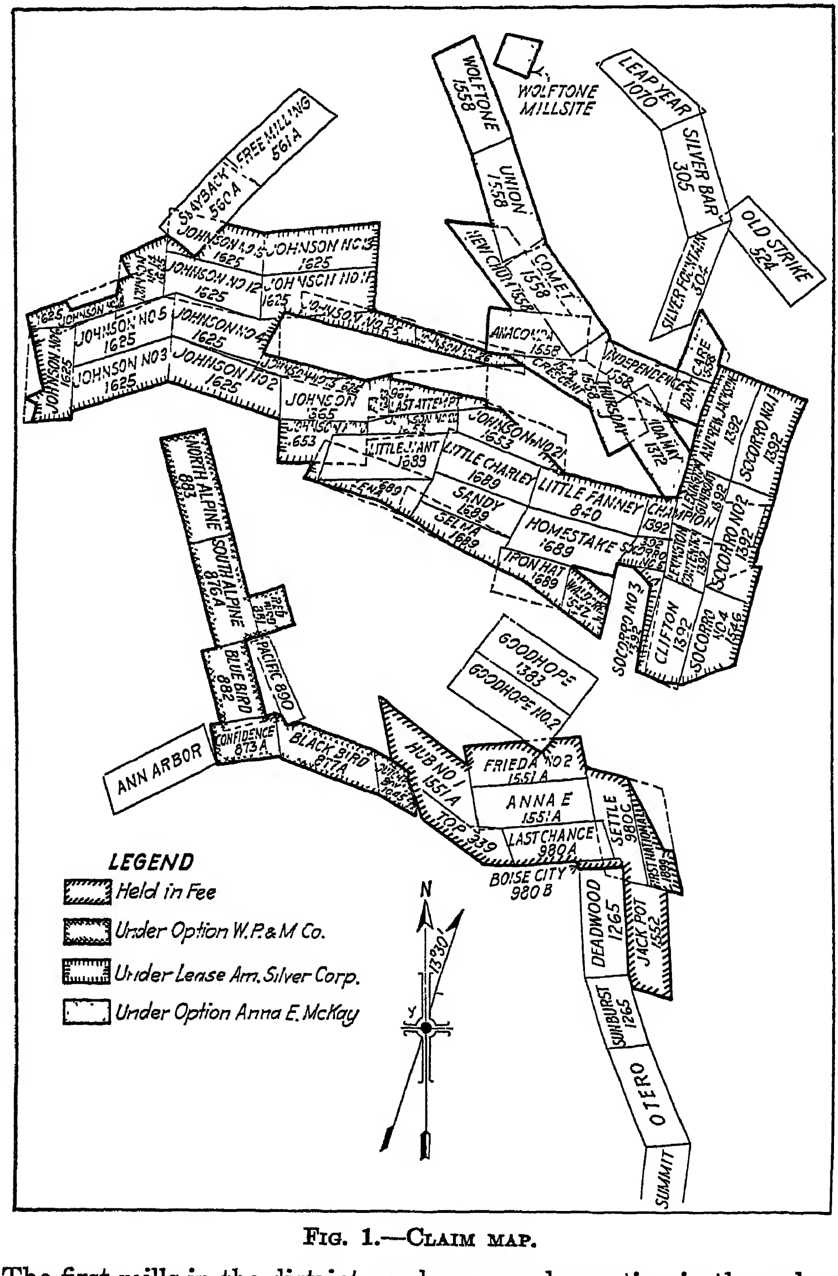

FIG. 1.—CLAIM MAP.

The first mills in the district used pan amalgamation in the reduction of their ores, probably with very low extraction. These were followed

by cyanide plants with sand and slime treatment or with fine grinding and all sliming, both with and without gravity concentration.

Geology of District

The regional rocks of the Mogollon district are principally a series of rhyolite and andesite flows of Tertiary age. Deep canyons have exposed a thickness of several thousand feet of these different flows, of which 1200 ft. has been mapped and described in detail by Ferguson.[1]

The area has been extensively faulted and many of the resulting fissures have been mineralized. The most productive veins in the district strike approximately N 70° W. The most important of these are the Last Chance-Confidence vein and the Fanney vein. Both are terminated on the east by the Queen fault, which has an average strike of N 10° E and can be traced across the entire district. Local variations in the strike of the veins are common with a decided change in direction near the Queen vein. The average dip of the principal veins is from 65° to 70° while many of the minor veins are in some places nearly vertical.

The outcrops of the veins are in most cases readily traceable, but, as pointed out by Scott,[2] they are not a true indication of the presence of ore deposits.

In general, oreshoots, where encountered, are large. Five oreshoots in the Last Chance vein have produced in the aggregate over 600,000 tons of ore. A vertical range of 300 to 500 ft. is not uncommon with a horizontal range usually somewhat less. In the Fanney vein, on the upper levels, one oreshoot was stoped for a length of 1500 ft. The width of the ore varies greatly. In the larger stopes, a width of 8 to 15 ft. is common, with 20 to 24 ft. about the maximum. Stoping widths of 2½ to 4 ft. are the rule in the minor veins.

In most cases where one wall rock is andesite, the opposite wall is rhyolite, but in a few instances both walls are andesite and in others both are rhyolite. Very little timbering or filling is ever required in stoping. Many stopes have stood for years after all ore was extracted without any signs of caving, and scarcely ever is there any danger of caving before all broken ore is drawn. Such conditions are especially suitable for shrinkage stoping, which is the method of extracting the ore that has been employed exclusively in recent years.

The predominant gangue minerals are quartz and calcite. Fluorite is common but is not characteristic either of the presence or absence of ore. Fragments of wall rock, especially andesite, are often present and

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[1] Henry G. Ferguson: The Mogollon District, New Mexico. U. S. Geol. Survey *Bull.* 715L (1920).

[2] David B. Scott: Ore Deposits of the Mogollon District. *Trans.* (1920) **63**, 289.

very frequently these fragments are surrounded by quartz and silver sulfides. Argentite and pyrite are the most prevalent of the valuable minerals. When alone, the pyrite is often barren, but when it is associated with argentite it generally carries gold. In high-grade ore, native silver and free gold are often observed, and frequently cerargyrite and bromyrite.

General Description of District

The rugged cliffs and deep canyons of the Mogollon Mountains, though tempered by the climate of the Southwest, remind one of the San Juan district of Colorado. The elevation of the principal mine surface plants is close to 7000 ft. To the west the range drops, at first abruptly, then more gently across the basin of the San Francisco River, to an elevation of 4900 ft. A few miles to the east, are the highest peaks in the Mogollon range, several being nearly 11,000 ft. above sea level.

The climate is very invigorating with little hot weather in the summer and seldom any very heavy snows, at the altitude of the mines, in the winter. In normal years the local streams supply sufficient water for milling and general purposes.

The veins, as a rule, make little water. The Last Chance fissure at the bottom of No. 3 shaft, at a depth of 1700 ft. below the outcrop, is open and drains any ordinary seepage or surface flow that may reach the mine workings.

As observed by Ferguson,[3] the Queen vein probably tends to divert ground water from the other veins to the west of it.

An adequate supply of mine timber is available in the Gila National Forest, in which the Mogollon district is located. In early operations there was a total disregard of reforestation, so that the mountainsides in the vicinity of the mines have been almost completely denuded of their stand of spruce and pine. With the present careful marking of mature timber for cutting, leaving the best seed trees to assure a good new growth, the supply should be ample for all future needs of the district.

Powder, which was formerly shipped mostly from Colorado, is now supplied from the plant of the Apache Powder Co. at Curtiss, Ariz. Aero brand cyanide is shipped from Niagara Falls, Canada.

De La Vergne engines, having a total sea-level rating of 1540 hp., have been installed in the camp. The engines develop approximately 78 per cent. of their sea-level rating at 7000 ft. elevation, making available a total of 1200 hp., or 900 kw. Fuel oil, which constitutes two-thirds of the freight hauled, comes either from Oklahoma or California.

Mexican labor is employed almost exclusively in the mines. Formerly considerable Italian labor was available for underground work but during

[3] *Op. cit.*, 192.

the war these men were attracted to the coal fields by the very high wages paid in those districts. The Mexican labor, particularly on contract work, is generally satisfactory. The district frequently suffers from an inadequate supply of miners, so that men often have to be brought in from the outside. The scale of wages is approximately the same as in Tyrone, Lordsburg, Fierro and Hanover. All classes of labor are on an 8-hr. schedule and the men are not unionized.

Transportation

The most important external factor affecting the cost of mining has always been the remote location of the camp, and the consequent high cost of transportation. For many years all of the hauling was done by team, but with the general improvement in road conditions and the bridging of the Gila River, trucks have entirely supplanted teams and have made possible a material reduction in transportation costs.

Table 1 shows the costs of the Mogollon Mines Co. in transporting 12,953 tons of freight a distance of 75 miles, during which period the company milled a total of 420,781 tons of ore.

TABLE 1.—*Transportation Costs of Mogollon Mines Co. on* 12,953 *Tons Freight*

	1914–1922	1922
Ore milled, tons	420,781.0	46,036.0
Total freight hauled, tons	12,953.0	1,484.0
Length of haul, miles	75.0	75.0
Total ton-miles hauled	971,541.0	111,318.0
Total cost of haulage	$319,673.07	$29,492.02
Cost per ton-mile	$0.329	$0.265
Cost per ton of ore milled	0.760	0.641

The company has always contracted its hauling, believing that minimum costs could be secured only by the close supervision of some one directly interested in the results obtained. In dry weather, with hard roads, 5-ton White trucks have shown the greatest economy; in wet weather, 2-ton White trucks on pneumatic tires have proved the most reliable; consequently, for all-year conditions a combination of the two sizes will work out to the greatest advantage. Welded steel tanks for fuel oil have proved much more satisfactory than riveted tanks.

Exploration and Sampling

The well-defined and prominent outcrops in the Mogollon district have been the initial guide in all exploration work. The veins have been developed either by inclined shafts sunk from the surface or from under-

ground levels, or by vertical shafts sunk in the hanging wall and cutting the vein at a depth of several hundred feet. Levels have usually been driven at regular intervals of 100 or 200 ft. with raises, winzes, and cross-cuts located, where possible, both to explore promising portions of the vein and to assist in extracting known ore.

Underground car samples are taken by trammers on each shift from every development face. Later, for purposes of preparing assay plans and estimating reserves, moil samples are cut usually at intervals of 10 ft.; in spotted or high-grade ore, at intervals of 5 ft. In winzes and raises, the samples are ordinarily cut at a difference of elevation of 5 ft., but on alternate sides of the raise or winze. Backs of stopes are cut at intervals of 10 ft., but on each successive slice the samples are cut half-way between those on the previous slice.

Assay plans are made on a scale of 1 in. = 40 ft., the same as the main working plans and elevations. Stope maps are made on a scale of 1 in. = 10 ft., and show, the width and value of the ore, where each cut was taken and in addition the tonnage and value of the ore broken in the stope during each month. The cumulative total of the ore broken, less the tonnage drawn, provides a monthly estimate of the broken ore remaining in the stope.

The average grade of ore is calculated from the foot-ounces gold and foot-ounces silver, a special loose-leaf record form being used for recording this information. In estimating the tonnage of ore in place, an allowance of 13 cu. ft. per ton is made. Moisture is quite uniform and amounts to 4 per cent. on average ore.

The sampling of the smaller blocks of ore generally checks closely with the tonnage and grade of ore produced but the larger blocks are rarely sufficiently developed ahead of mining to permit more than rough estimates of their probable production. As stoping proceeds and the width and grade of ore are more clearly established, it has been found that the monthly estimates of ore broken, when finally checked against the ore drawn, agree closely both as to tonnage and grade. The ore drawn, however, commonly exceeds the estimates of the tonnage of ore broken while the grade of the ore drawn will be correspondingly less, an experience common to almost all sampling when the limits of the ore are not sharply defined.

In addition to the sampling of development faces and estimating the mineral contents of orebodies, chute samples are taken and the tonnage drawn from each chute is estimated by the number of cars trammed. The average assay in ounces gold and silver is calculated for each chute at the end of the month. As the ore is trammed to the mill, the cars are weighed and a grab sample is taken from the top of each car and a composite sample made for each shift. At the end of the month, the calculated average of the scale samples is checked against the chute samples

taken underground and against the average mill heads as determined from the ounces gold and silver in the bullion, concentrates and tailings (production plus tails). On account of the slightly higher assay of the fines in the ore, the average mine sample and the scale sample are usually slightly higher than the mill head sample. The yearly averages, however, seldom show a variation of more than 10 per cent. compared with the mill heads and generally agree within 3 or 4 per cent. On spotted and high-grade ore, the underground car sampled become very unreliable. To be used at all, lots of ore after being sampled underground should be crushed and checked in a Snider, Vezin or other automatic sampler.

Table 2.—*Comparison of Mine and Mill Sampling*

Weight of ore, tons	51,862	43,993
Mill average	$11.084	$10.254
Scale sample	$11.218	$11.193
Per cent. above or below mill sample	1.20	9 17

Early Mining Methods

In the early mining operations in the Mogollon district, the ore was extracted nearly as fast as broken. Levels were close together and stoping was frequently carried on from platforms carried on light stulls. The backs of drifts were generally timbered, leaving no pillars between chutes. At one time in one of the mines, levels were driven 35 ft. apart and the ore was broken by overhand stoping, all broken ore being shoveled. The isolation of the district and high operating costs made necessary cheaper methods of working and facilities for handling somewhat larger tonnages. Though the methods that have been worked out by different operators in the district are here employed in handling comparatively very small tonnages, they have been sufficiently standardized so that they can be easily expanded to handle several times the quantity required. An adequate supply of cheap power from an outside source, the substitution of mechanical haulage for hand tramming and mule haulage, with increased drilling and hoisting facilities are the principal factors that would require attention.

Present Mining Methods

As has been mentioned, the wall rocks of the veins in the Mogollon district are rhyolite and andesite. They stand well without timbering. The dip of the veins is generally close to 70°, eliminating any difficulty in drawing off broken ore. In the larger orebodies, a stoping width of 8 to 15 ft. is usual, with 20 to 24 ft. the maximum. The grade of the ore is fairly uniform, with occasional enrichment. Some waste occurs in the stopes but the lean portions of the vein are generally on the foot wall

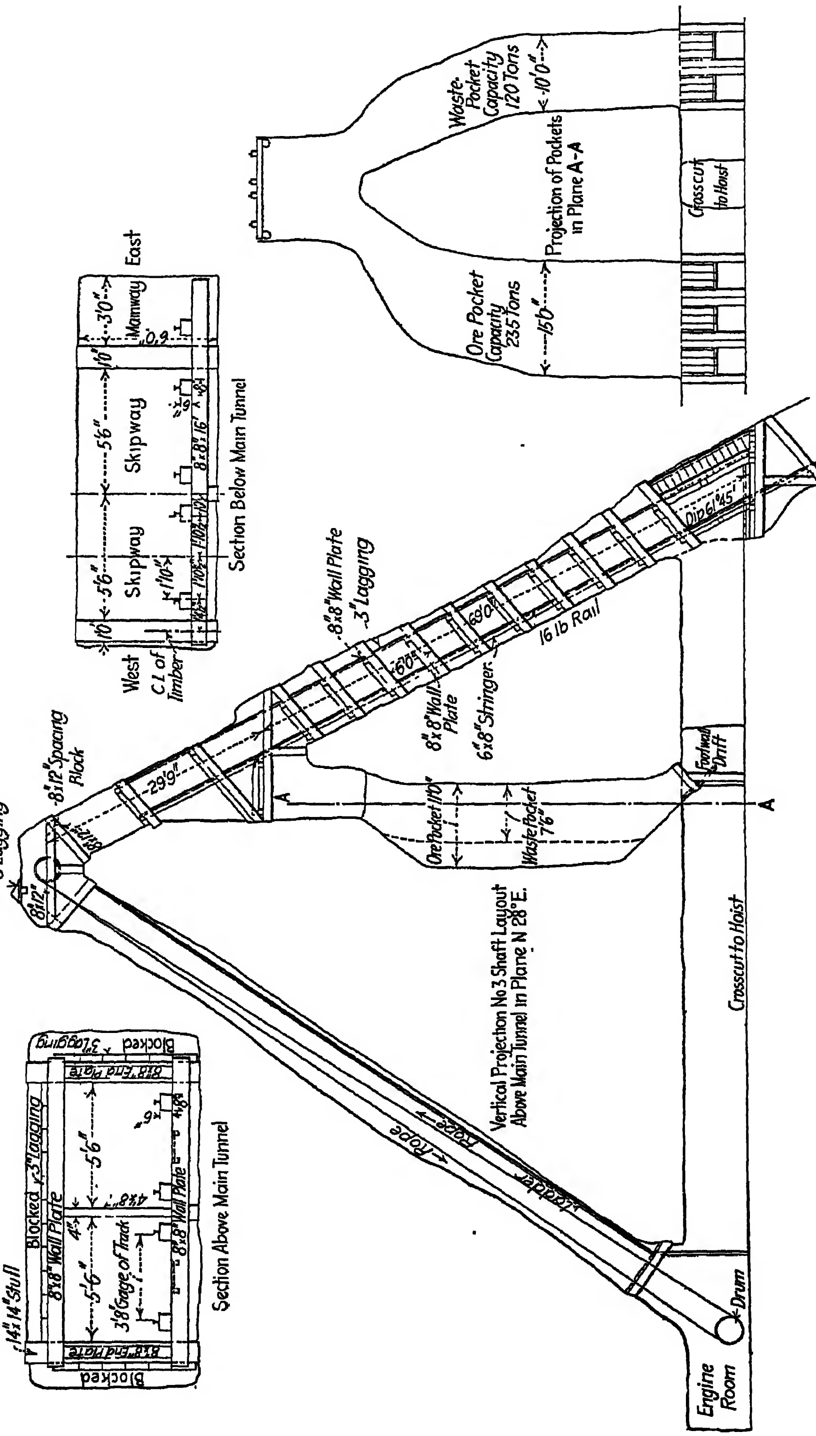

FIG. 2.—GENERAL LAYOUT AND TIMBERING OF NO. 3 SHAFT.

or hanging wall and can be left in place. The ore is hard and contains a low percentage of moisture.

Shrinkage stoping is doubtless the best and cheapest method of mining under such circumstances, the only disadvantage being that approximately two-thirds of the broken ore cannot be drawn until the stope has been completed. This feature is not entirely unfavorable, however, as it necessitates more diligent search for other ore in order to maintain a stated output.

Mine Openings, Shafts, Tunnels

The Last Chance-Confidence vein has been developed from underground hoisting stations on the main haulage level used in tramming ore to the surface. This level is approximately 700 ft. below the highest portion of the outcrop and has a total length of 4087 ft. In 1915, the Mogollon Mines Co. began the construction of a new three-compartment inclined shaft which was sunk to a depth of 980 ft. below the main haulage level. Details of the shaft layout are shown in Fig. 2. The size of the shaft outside of timbers is 16 by 6 ft. The two hoisting compartments are 5½ by 4½ ft. in the clear, with a ladderway 3 by 4½ ft. in the clear. The shaft timbers are treated with carbolineum wood preservative. The shaft was located 1550 ft. from the mouth of the haulage adit and approximately under the highest part of the outcrop. The horizontal rails of the skip dump are 69 ft. on the incline above the haulage level. By means of a balanced gate, hinged in the middle like a large butterfly valve, skips dump either into the ore or waste pocket. The former has a capacity of

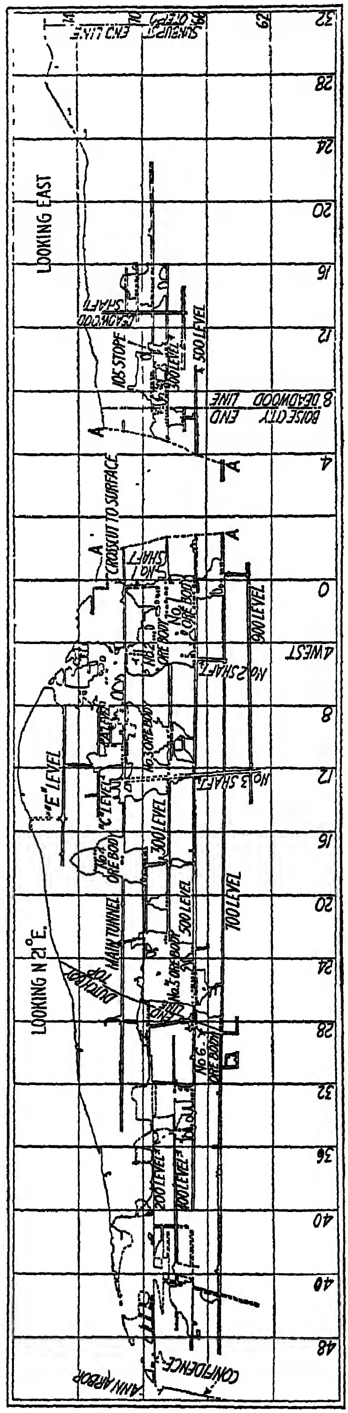

Fig. 3.—Vertical projection of Last Chance-Confidence and Deadwood mines.

250 tons and the latter of 125 tons. There are three loading chutes for ore and two for waste, located in a foot-wall drift 30 ft. from the main drift. Standard arc gates with stopper boards for the fines are used in the chutes. Ore is delivered to the mill in trains of 10 cars, the cars being the same size throughout the mine. Mule haulage is used.

The Fanney mine is equipped with a three-compartment vertical shaft, having two hoisting compartments 4 ft. by 4 ft. 4 in. and a manway and pipe compartment 3 ft. by 4 ft. 4 in. The collar of the shaft is concreted for a depth of 10 ft. and is timbered below this point. The shaft has a 75-ft. steel headframe and skips discharge into bins feeding the coarse crushers. Ore from the crushers is delivered by belt conveyor to the mill bins.

Underground Development Plans

The vertical projection of the Last Chance-Confidence vein, Fig. 3, shows the development for a little over 1¼ miles along the strike of the vein. The levels are principally 200 ft. apart, except in the old Confidence and Deadwood workings, where they are 100 feet.

The main drifts are driven on the vein and, after cutting an orebody and determining its length, a raise, either with two chutes and a manway between or with a single chute and manway, is driven in the ore to the level above. The raise serves to prospect the ground between the levels, aids ventilation, and is needed in handling supplies for the stope. Raises for the chutes are put up on the foot wall at intervals of 20 ft., if the ore is continuous. Details of the chutes are given in Fig. 4. Pillars of ore, averaging 15 ft. wide, are left between the chutes, the tops of the pillars being trimmed off to form a hopper above each chute. Where a stope is later coming up from below, the present practice is to drive a foot-wall drift about 15 ft. from the main drift so as to preserve a tramming level around the orebody. The pillars of ore between the chutes and the ore in the bottom of the original drift are then broken into the stope as it comes up from the level below.

Usually, sublevels are not required unless the development has shown that the ore is not continuous between the main levels.

In the development of narrow veins, not over 4 or 5 ft. wide, no pillars are left between the chutes, as a rule, the broken ore being carried on stulls and lagging instead.

In 1922, the average cost of 4904 ft. of development, of which 73 per cent. was drifting, was $12.10 per foot.

Tracks are 18-in. gage and are run on a grade of 0.6 per cent. A level board 100 in. long, with the grade set off on one end by means of a set screw and lock-nut, has been found most useful in assisting trackmen to carry a uniform grade. Twelve-pound rail is used in all development work, with considerable 16-lb. rail on the main haulage level to the mill.

The main air line entering the mine is 6-in. pipe with 2½ in. on the levels, and 1½-in. and 2-in. in raises and stopes. Sufficient water for filling the tanks of Leyner machines can usually be obtained from sumps near where the drilling is going on.

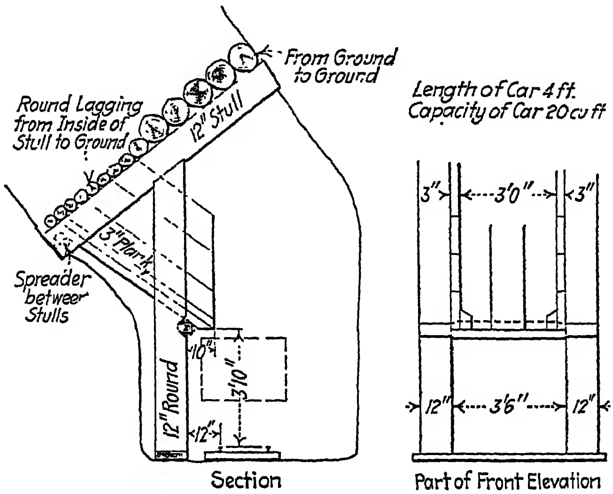

FIG. 4.—STANDARD TIMBER CHUTE.

MINING PRACTICE

Drilling and Blasting

The drifting machine in general use in Mogollon is the Ingersoll-Rand No. 248 Leyner, using 1¼-in. hollow round steel, with double-taper cross bit. The CC-11 machine is in general use for stoping, having in one test shown 45.5 per cent. increase in drilling speed over the older BC21 machine. The only stoper steel in use at present is ⅞-in. quarter-octagon, electric-furnace steel made by the Midvale company, which has shown the smallest breakage of any steel ever used. The double-taper cross bit is also used on stoper steel. Jackhamers in common use are the BCR430 and the RA-10 of the Ingersoll-Rand company, using ⅞-in. hollow hexagon steel.

A vertical bar is used in drilling drift rounds. The ground would, in general, be classed as hard and tight, so that a comparatively short round is drilled which breaks approximately 3½ ft. Generally 14 to 16 holes are required in drilling a down-cut round. In stoping, a vertical face, from 3 to 5 ft. high, is carried across the back of the stope and holes are drilled at a slight angle to the face. Leyners are never used in stoping except in rare cases when extremely hard ribs of ore are encountered.

The explosives generally used are 30 and 40 per cent. gelatin; the former is being used in stopes and moderately hard raises and the latter

in drifts and crosscuts. The 33 per cent. ammonia powder has been used quite extensively in stoping, but 30 per cent. gelatin is easier to load and usually breaks the ground as well. No. 8X blasting caps are used exclusively with either Black Monarch or Sylvanite fuse. Delay electric igniters have been found especially satisfactory in removing old shaft pillars and in sinking.

Tamping has not been as generally used in Mogollon as it was when operations were more concentrated. Details of a tamping bag filler and notes on the use of tamping in Mogollon were published in 1916.[4]

"Bulldozing," as a general rule, is not required except in the widest stopes and then only when cleavage planes have a tendency to cause the ore to break down in slabs. Ordinarily, a small amount of hammer work will keep the chutes free from big boulders.

Drifting and Stoping

Drifts average 5 by 7 ft. in section. Two men are used on a machine and drilling is always done from a vertical bar using an 18-in. or 24-in. arm. When possible, drilling is done on one shift and mucking on the opposite shift, in order to give the machine men a clean set up every day. When necessary to drill on both shifts, shoveling back and setting up takes from one hour to one and one-half hours.

An advance of 3 to 3½ ft. per machine shift is customary, the maximum being 5 to 6 ft. in most favorable ground. No timbering, as a rule, is required in the drifts.

The tons of ore extracted or broken per foot of development shows considerable variation in the district. On the Last Chance-Confidence vein, the records of the Mogollon Mines Co. over a period of nine years show 412,631 tons of ore broken for 25,842 ft. of development, or an average of 16.0 tons per foot of development. Scott[5] reports the following averages in other mines of the district: Fanney, 10.5 tons per foot; Deadwood, 11.8 tons per foot; Pacific 1.0 ton per foot.

There is little variation in the methods of carrying up shrinkage stopes except in the case of the manways. In some instances, the original raise in the orebody does not go up to the level above, in which event a manway is required on each end of the stope with a door in the drift between the manways, which can be closed so as to force a current of air up one manway, through the stope and down the other manway. This method is suitable only when there is good ventilation on the level from which the stope is started. In most cases where the original raise through the ore goes to the level above, a manway carried up on either end of the stope affords ample ventilation. As a matter of convenience, in a long stope a manway may be carried up on each end until the stope is well swung,

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[4] *Min. & Sci. Pr.* (Sept. 9, 1916). [5] *Op. cit.*, 309.

after which men and supplies enter from the level above and one manway from the level below furnishes all the ventilation required.

As stoping operations are generally started on one end of an orebody while raises are being put up on the other end, the back of the stope and the broken ore may, at first, have the appearance of one side of a rill stope. As stoping progresses the back is carried more nearly horizontal. Nearly vertical holes are drilled in a row across the vein at right angles to the strike, carrying a face 3 to 5 ft. high according to the character of the vein matter. The tons of ore broken per drill shift varies from 16 tons in the case of the wider stopes to 8 tons or less in the narrow stopes. The ore broken per pound of explosive shows a corresponding variation, averaging from 1 ton to ½ ton. In order to avoid using staging, the ore should be kept within 3 or 4 ft. of the back on the foot-wall side of the stope, keeping careful watch to see that no ore is left on either wall.

Timbering

One of the favorable factors in mining operations in the district is the very small amount of timber required. Almost all openings stand well without support so that very little timber is required except for shafts and manways and the chutes in the stopes.

Red fir and spruce are the only native timbers satisfactory for underground use. Juniper is usually too small and irregular to be suitable. Sawed timber costs $50 per M. ft. B. M. and round timber 2¼ cents per inch-foot; *i. e.*, a round timber 10 in. in diameter costs 22½ cents per running foot. C. A. Botsford, mine superintendent, has prepared Table 3, which shows the comparative costs of sawed timber and round timber of approximately the same area.

TABLE 3.—*Comparative Costs of Sawed and Round Timber*

Sawed Timber			Round Timber		
Dimensions Inches	Area, Sq. In.	Cost for 10 Ft.	Diameter, Inches	Area, Sq. In.	Cost for 10 Ft.
3 × 8, 4 × 6	24	$1.00	5.5	23.76	$1.24
6 × 6	36	1.50	6.75	35.78	1.52
8 × 8	64	2.67	9.0	63.62	2.03
10 × 10	100	4.17	11.25	99.40	2.53
12 × 12	144	6.00	13.5	143.14	3.04

From this table, it can be seen that, at the prices given, a 12 by 12-in. sawed timber costs nearly double the amount of a round timber of equal area; a 6 by 6-in. sawed timber, however, costs almost exactly the same as a round timber of equal area.

All permanent timbering is given a brush treatment with carbolineum, which has been found very effective in preventing dry rot.

Tramming and Haulage

All mine cars used in the Last Chance-Confidence workings, with few exceptions, are 20-cu. ft. capacity, 18-in. gage, end-dump, equipped with Hyatt roller bearings. Mule haulage is used on the main tunnel, 500 and 700-ft. levels. Adverse factors affecting haulage costs are small tonnage handled, long distances trammed, and widely separated working

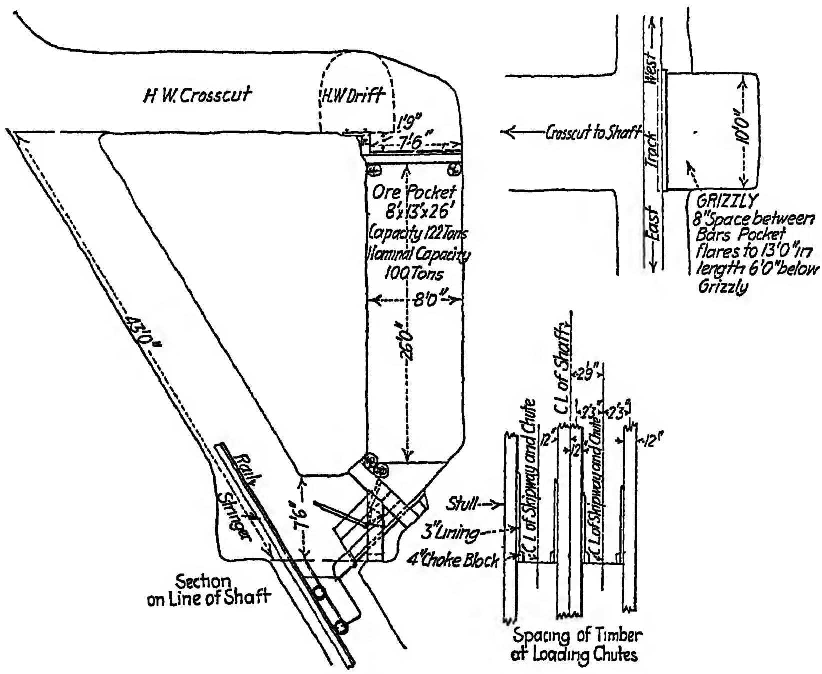

FIG. 5.—ORE POCKET AND LOADING CHUTES.

faces. Twelve-pound rail is used in the main drifts and 12 and 16-lb. rail on the main haulage level. All new work is laid out with 0.6 per cent. grade.

Underground Storage and Dumping

In the Last Chance mine, underground storage pockets are designed to hold approximately 125 tons, Fig. 5. The cars are dumped by hand on to a grizzly dropped 3½ ft. below the level of the track. The present practice is to make the grizzlies out of discarded stamp stems, using a 6-in. spacing. With a grizzly 10 ft. long, three cars can be dumped without moving the train. Waste is either dumped into old stopes or directly into the skips for hoisting to the surface. Skips are

loaded from the ore pocket direct without the use of a measuring cartridge. Handles for the arc gates and electric signaling switches are conveniently arranged for the operator between the two skipways.

Hoisting

Both the Last Chance and Fanney mines are equipped with double-drum electrically driven hoists made by the Denver Engineering Works, the former of 5000-lb. rope pull capacity and the latter of 10,000-lb. Both hoist in balance, and use skips holding 1½ tons of ore.

The equipment of the Mogollon Mines Co. was designed to hoist 200 tons in 8 hr. from the 500-ft. level. Low hoisting speed is used to reduce the power required to the minimum. The motor used is 40-hp., 440-volt, three-phase, 60-cycle slip-ring type. Hoisting ropes are ¾-in. 6×19 regular lay. The life of the ropes has been materially increased by using idlers in the shaft, equipped with Hyatt roller bearings.

The hoist at the Fanney mine uses an 80-hp., 550-volt d.c. motor operated from a belt-driven generator located in the main power plant. Skips and cages, which can be used when required, are equipped with safety dogs. Portable hoists in general use are the Ingersoll-Rand Little Tugger with single drum and the Sullivan Turbinair.

Pumping

As the mines make little water, almost no pumping is required. A bulkhead on the 500-ft. level of the Fanney mine cuts off water from the Queen vein, which at times builds up a pressure of 120 to 125 lb. per sq. in. The accumulated water has been pumped out by a 4 by 8-in. triplex pump having a capacity of 72 gal. per min., running 16 hr. a day, in three weeks, indicating a volume of about 1,450,000 gal. The Queen vein, where cut on the 700-ft. level of the Last Chance mine, was dry.

Ventilation

The mines on the principal veins in the district have been connected underground so as to afford good natural ventilation. Doors are used to direct the air currents through the workings when necessary.

Lighting

Stations and main haulage levels in the Last Chance mine have electric lights supplied from 220-volt a.c. circuit. There is no lighting provided in the shafts. Miners use No. 3 Wolf carbide lamps.

Telephones and Signaling

Standard underground telephones are used on the stations and can be connected with the surface telephone system. Lead-covered telephone wire is used exclusively underground.

The mine-shaft signal system employs three distinct modes of communicating between any station or loading pocket and the hoist engineer.

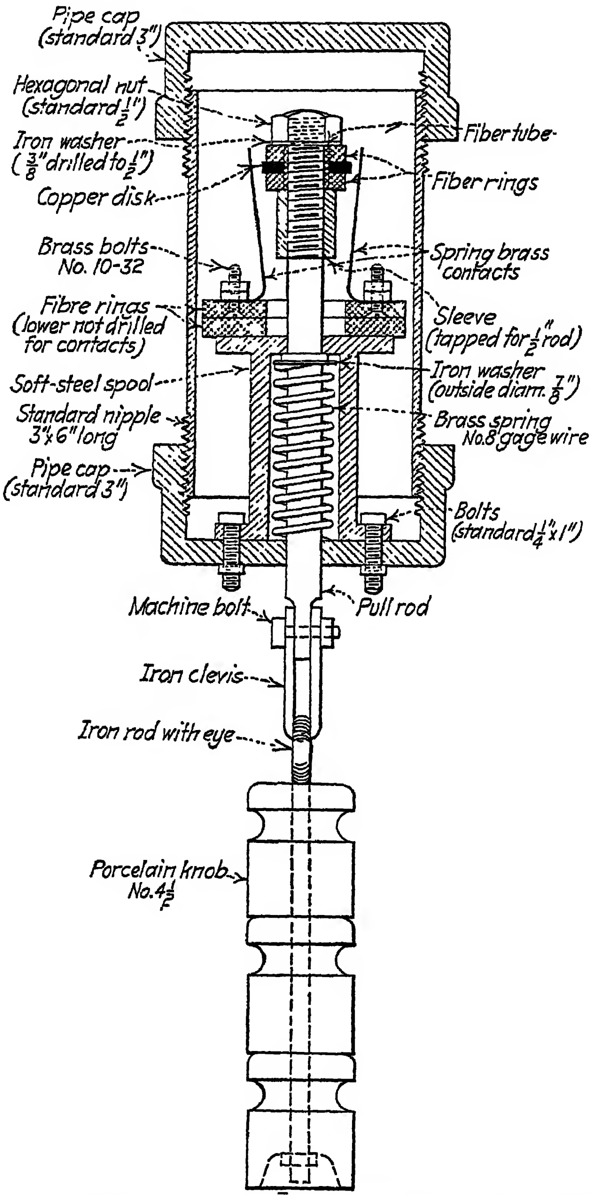

FIG. 6.—ELECTRICAL PULL SWITCH, MINE SIGNAL SYSTEM.

For calling a skip to any station or loading pocket, a flash system is used, indicating to the engineer, to the cager and on all of the stations, the level on which the cage is wanted.

Beside the hoist engineer is a solenoid gong for each compartment in the shaft. A pull switch is placed at each station and loading pocket, for operating the gong, and the skips are moved only when the gong or bell signal system is used. Details of the pull switch, which is the same for flash signals and bell signals, are given in Fig. 6. The pull switch is very rugged, all of the connections being inside a standard 3-in. pipe. It can be struck with a hammer or have a stream of water directed on it without sustaining any damage.

The design of the solenoid is shown in Fig. 7. A valuable feature of the solenoid is that it operates with a certain amount of lag so that no matter how fast the pull switches are operated the signals to the engineer are clear and distinct.

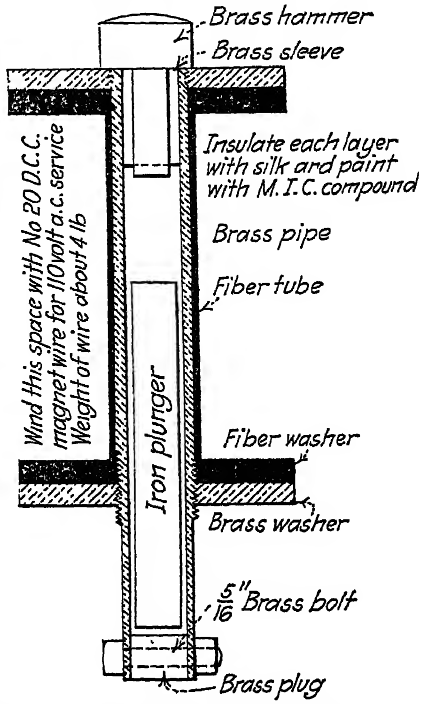

Fig. 7.—Solenoid for electric signal system.

The flash and bell signals operate from the 220-volt lighting circuit. In a wet shaft, it might be advisable to step the voltage down to 110 or even to 32 volts and have the solenoid redesigned for the lower voltage.

The cables for the flash and bell signals are each three-conductor, No. 14 copper, insulated with 3/64-in. rubber covering and single braid, around all of which is a 1/16-in. lead sheath, armored with a double taping of band steel between two wraps of asphalted jute. A two-conductor cable of the same specifications is used between the main cables and the pull switches, using welded steel junction boxes instead of the conventional and more expensive cast-iron boxes.

A flexible steel bell lina is placed in each compartment of the shaft for signaling to the hoisting engineer from any point in the shaft.

Unit Cost and Production

Unit costs and production for the year 1922 are given in Tables 4 and 5.

TABLE 4.—*Unit Costs, Year of* 1922

Based on 50,144 tons broken ore and 4904 ft. development work

	Ore Breaking		Development		Total Mine Operating (Ore Breaking and Development)	
	Per Ton	Per Cent.	Per Ton	Per Cent.	Per Ton	Per Cent.
Drills and drilling	$0.650	23.66	$0.463	39.21		
Air lines	0 053	1.93				
Explosives	0.427	15.54	0.267	22.59		
Timbering	0.275	10 00	0.084	7.14		
Tramming	0.633	23.06	0.244	20.65		
Hoisting	0.247	8.99	0 054	4.50		
Pumping	0.006	0.20				
Assaying and surveying	0.088	3.18	0.002	0.14		
Tools and sundry supplies	0.002	0 10				
Illuminants	0 034	1.24	0.015	1.24		
Tool sharpening	0.133	4.85	0.054	4.53		
Surface	0.045	1.64				
Superintendent and shift bosses	0.154	5.61				
Total	$2.747	100.00	$1.183	100.00		
Summary:						
Labor	$1.683	61.30	$0.682	57.60	$2.365	60.20
Supplies	0.731	26.60	0.378	32.00	1.108	28.20
Power	0.332	12.10	0.123	10.40	0.457	11.60
Total	$2.747	100.00	$1.183	100.00	$3.930	100.00

The different classes of development work, with their costs, are as follows:

	No. Feet	Cost Per Ft.
Drifts	3555	$11.65
Raises	877	14.08
Crosscuts	183	11.86
Winzes	289	11.66
Total	4904	$12.10

All ore breaking is done by day labor. Development work is usually contracted and paid for by the foot of advance. Tramming is both by day labor and contract.

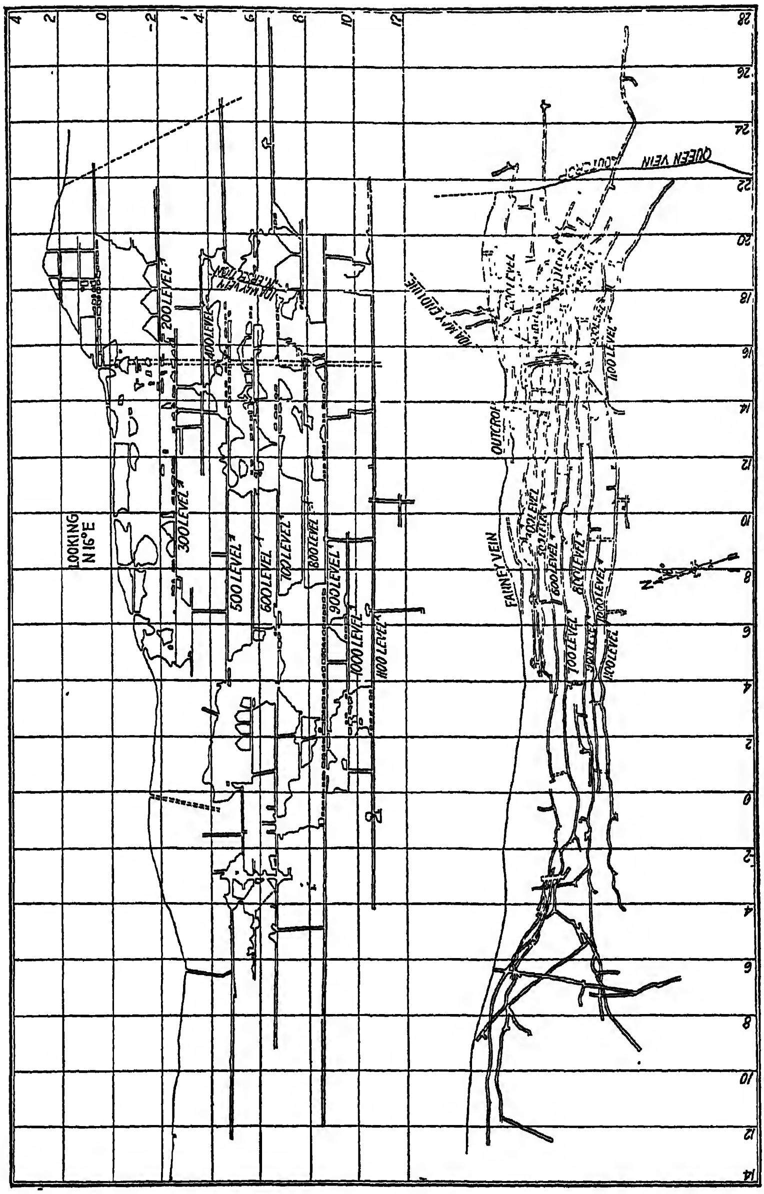

Fig. 8.—Vertical Projection and Plan, Fanney Mine.

TABLE 5.—*Unit Production, Year of* 1922

Based on 10.86 tons ore broken per drill shift

	In Stopes	All Underground Labor[a]	All Surface Labor[b]	Entire Organization[c]
Tons per man per hour..........	1.36	0.23	0.43	0.145
Man-hours per ton............	0.74	4.35	2.31	6.92

[a] Including steel sharpener, and portion of blacksmith and helper.
[b] Exclusive of office, but inclusive of mill and power-plant labor
[c] Including office force, also mill and power-plant labor.

EXPLOSIVES

The average consumption of all grades of explosives for the year 1922 was as follows:

In stoping............................	1.58 lb. per ton ore broken
In development.......................	10.60 lb. per foot advance
In all mining operations................	2.62 lb. per ton ore broken

COMPRESSED AIR

The principal air compressors in use are the four described in Table 6.

TABLE 6.—*Air Compressors Used by Mogollon Mines Co.*

	Make and Type	Size, Inches	Drive	Capacity, Cubic Feet
1.	I. R. Imperial XB2	21 × 12 × 16	150-hp. motor, 220 volt	1085
2.	I. R. Imperial XB2	13 × 7½ × 12	75-hp. motor, 220 volt, short belt	385
3.	I. R. Class P	14¼ × 9¼ × 14	Direct connected to 90-hp. De La Vergne engine (sea-level rating)	457
4.	I. R. XB2 Class 10	20 × 11 × 16	Belt from counter shaft	984

POWER

As noted in the general description of the district, all power used is furnished by De La Vergne engines, the fuel oil being hauled 75 miles from Silver City in trucks. On account of the importance of power, especially in isolated operations, the power-plant costs for the year 1922 are given in Table 7.

TABLE 7.—*Power-plant Costs of Mogollon Mines Co. for Year 1922*

	TOTAL	PER CENT.
Engineers	$ 6094.76	11.11
Oilers	4172.47	7.61
Miscellaneous	3313.23	6.05
Total labor	$13580.46	24.77
Fuel oil	32593.10	59.44
Kerosene or tractor oil	163.32	0.32
Lubricating oil	2453.35	4.47
Waste	219.82	0.40
Electrical supplies	1443.55	2.63
Renewals for De La Vergne engines	2968.13	5.41
Miscellaneous	1403.29	2.56
Total supplies	$41244.56	75.23
Total labor and supplies	$54825.02	100.00

Total cost of power	$54,825.02
Total kw.-hr. at switchboard	1,894,460
Cost per kw.-hr. at switchboard	$0.0289
Total b.-hp. hr.	3,063,928
Cost per b.-hp. hr.	$0.0179
Pounds fuel oil per kw.-hr.	0.8865
Pounds fuel oil per b.-hp.-hr.	0.5481
Total fuel oil used, pounds	1,679,335
Cost of fuel oil per pound	$0.0194
Cost of fuel oil per gallon	0.1436
Cost of oil per barrel Mogollon	6.03

COMPARATIVE COST OF SUPPLIES

As most of the operating costs that have been given in this paper refer to the year 1922, it is interesting to note that the comparative cost of supplies, based on July 1, 1914, as 100, was 160 at the beginning of 1922 and 150.2 at the end of 1922. The price of 12 commodities enters into the index: powder, fuse, caps, drill steel, battery steel, cyanide, lead acetate, sheet zinc, belting, fuel oil, lime, and native lumber. The price of all except the last two items was taken at Silver City. The lime and lumber are of local origin.

MEDICAL ATTENTION

A doctor and emergency hospital are maintained in the district, for which all married men contribute $2 per month and all single men $1.50 per month.

Mining Methods of the Telluride District

By Charles N. Bell, Denver, Colo.

(New York Meeting, February, 1924)

The Telluride mining district of southwestern Colorado is defined by the 37° 45′ and 38° parallels of latitude and 107° 45′ and 108° meridians of longitude.

Telluride was never a boom camp, but has had a steady growth. In recent years, notwithstanding the business depression of 1920, the business has been the same as in normal years. The first prospector is said to have appeared in 1875, in which year locations on the Smuggler vein were made. Shipments were made in 1877 and 1878, and in 1879 milling the ores was attempted. While the district produced high-grade ores, it did not become important until 1890, when the Rio Grande Southern R. R. was completed from Ridgway to Telluride. The railroad has since been extended to Durango.

In the fall of 1896, 400 stamps were in operation, and the annual production was $3,000,000.00, two-thirds of which was gold. The total production from 1875 to 1923 was: gold $58,274,527; silver 41,830,920 oz., valued at $31,016,893; copper 15,258,866 lb., valued at $2,533,291; lead 166,056,484 lb., valued at $8,324,910; zinc 18,141,182 lb. valued at $1,323,787; or a total production of $101,473,408.

All claims are held in fee. Most of them are 1500 by 300 ft. but claims located within the last few years are 1500 by 600 ft.

General supplies are obtained at Telluride, Denver, and Salt Lake. The largest mines are within 6 miles of the town of Telluride. Denver is 422 miles and Salt Lake is 439 miles, by rail, from Telluride. One-half of the distance to Denver and one-sixth of the distance from Telluride to Salt Lake is by narrow-gage railroad.

The labor is efficient. The best miners are Italians and Austrians, then Slavs, Greeks, and Mexicans follow in the order mentioned. There are no labor unions. There are no serious mine pumping problems, the drainage being through adits. The Tomboy and Smuggler companies pump water from their reservoirs, the former for milling and domestic purposes and the latter for milling, power, and domestic purposes.

Bibliography

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Geology

The ore deposits occur in filled fissures. The vein filling necessarily followed the fissuring which has been shown to be later than any of the rocks, including the latest lavas. The last eruptions occurred in the late Tertiary, and it is probable that the fissuring and ore filling closely followed the eruptions. The highest member of the volcanic series is rhyolite; next are alternating series of andesitic and rhyolitic lavas; under this is andesitic breccia, which occurs in a stratified arrangement supposed to have been caused by deposition of these materials in a lake. A coarse conglomerate containing boulders of granite, quartzite, schist, gneiss, limestone, and sandstone is next in the series. Below the conglomerate are shales, sandstones, limestones, and quartzite, but these are of no interest at this time as these beds have not been explored in connection with the principal veins of the camp. The Contention vein, contained in sandstone below the volcanic series, was operated on a considerable scale in 1900–1902. The ore has been found in the alternating bands of andesite and rhyolite and in the andesitic breccia.

The oreshoots are long, of fair depth, and narrow width. The longest oreshoot in the district is that of the Smuggler-Union-Humboldt vein, which has been stoped continuously for 8300 ft. with the possibility that it will be mined continuously for 13,000 to 15,000 ft. The possibility is mentioned because a barren section exists between the Humboldt mine and the adjoining mine to the north, called the Mountain Top, at the horizon where the workings are closest. The vein is continuous and the ore may exist at the other horizons. In 1904, T. A. Rickard reported the Smuggler-Union oreshoot to be the longest continuous oreshoot in the world; whether this is true today, I am unable to say. The walls range from good to poor. In general, the high-grade ores are near the surface, the grade diminishing to unprofitable ores in the lower portions of the andesite breccia; some small high-grade gold shoots, however, persist from the surface to the lower breccia and some high-grade orebodies have been found in the lower breccia.

The topographic features are striking. Deep cañons and steep ridges with sharp and precipitous peaks are prevalent. The elevation changes from 9200 ft. to nearly 13,400 ft. within 2 miles. So steep are the slopes of the ridges that the direct action of the sun's rays is shut off from the northward-facing slopes for a large part of the day. As a result, the early falls of snow accumulate on those slopes and in the basins below much more rapidly than on the slopes that face the south. The snow fall is so heavy that practically all work is carried on under cover.

Exploration, Sampling, and Estimating Methods

The exploration work is done by crosscutting to the veins and drifting on them. Diamond drilling is used at the Smuggler and Humboldt mines to determine if veins exist, but it gives no reliable data as to the value of the vein. There are some shafts in the district.

The ore has quartz gangue carrying varying quantities of sulfides, mostly pyrite, galena, and sphalerite. The precious metals are free gold, native silver, freibergite, polybasite, proustite, pyrargyrite, and stephanite. There are also chalcopyrite, sphalerite, mispickel, magnetite, and stibnite. The gangue minerals, besides quartz, include sericite, biotite, chlorite, amphibolite, apatite, garnet, orthoclase, kaolinite, calcite, siderite, rhodochrosite, dolomite, fluorite, and barite.

In the Lennox[1] scale of hardness the Smuggler ore is 0.94, the Liberty Bell 0.83, and the Humboldt 0.80. For comparative purposes, the Calumet and Hecla jig tailing is 1.33, Homestake ore 0.63, and Ray Consolidated ore is 0.37. Portions of the ores are friable and portions are tough. The moisture content of the ore, in places, is quite low.

Cut samples are taken systematically and averaged geometrically. Assays are recorded on assay plans. Some companies use a color scheme to record the assays: for example, non-pay ore may be recorded in black; pay ore in red; and ore above a predetermined amount, say $10 per ton, in yellow. The assay plan then shows, at a glance, the trend of the ore-shoots. Chute samples are taken if mill heads fall below a predetermined point. If the veins are narrow, the assays are reduced to a minable width.

The estimating of the orebodies is simple. The length, height, width, and dilution are the factors. The ores vary from 11¾ to 13 cu. ft. per ton. The tonnage mined is usually more than that estimated; the overrun is sometimes as high as 40 per cent. At one mine, it was found that all assays over $20 should be eliminated in estimating the ore; at another, when mining an ore containing coarse gold it is necessary to include all the high assays and even then the recovery is

[1] Grinding Resistance of Various Ores. *Trans.* (1919) **61**, 244.

more than estimated. When mining fine-grained gold ores and silver ores, all high samples should be eliminated unless supported by additional adjacent high samples. The sampling interval is usually 10 ft.

Mining Methods

Filled Stope

The old Smuggler vein of the Smuggler-Union Mining Co. was mined by the filled-stope method. The vein has a dip of 79° and a width of 24 to 30 in. The levels are 100 to 250 ft. apart, the walls are generally strong, though weak in places. The drift on the vein is back-stoped and timbered, either by stulls or square sets. The lagging is placed, and stulled or cribbed manways are started, two to the stope, or at 200-ft. intervals in long stopes. Chutes are placed every 50 ft. A 30 to 50-ton ore pocket is built over the chute, and mill holes, leading from the pocket up through the fills, are built from waste rock; one of the walls is used for a side of the mill. These mills were started off the pocket 24 by 24 in. Toward the top of the stopes, however, the chute builders are apt to get the chutes only 18 to 20 in. wide. The dimension perpendicular to the wall varies from 24 to 48 in. In order to get as uniform a slope as possible, and still take advantage of the hanging wall for one side, the holes were sometimes 3 and 4 ft. long; these were apt to give trouble through caving. The 14 to 18-in. square holes resulted in an excessive number of blocked chutes; 24 by 24 in. is standard. One good chute builder will raise two chutes a shift, of 8 hr., 3 to 4 ft. each, and block around them, unless there is no suitable rock convenient and he has to pack it or use wood blocks. End timbered mill holes are used when the waste rock is soft and unsuitable for building mill holes. A light round is first shot on the lagging; after this there is no danger of breaking the lagging.

The vein material is broken down and the pay ore mucked or shovel sorted into the mill holes; the waste material and material below pay grade are left in the stope for fill. Any additional fill necessary to bring the top of the muck within working distance of the back is shot from the foot wall. The muck is leveled off and a floor made before the next round is shot. Vein material may be shot directly on this floor, or on boards, canvas, or steel sheets.

This method was adopted, primarily, because shipping ore could be produced as well as milling ore. Ore of shipping grade can be sacked in the stopes or placed in a separate chute and the mill ore shoveled to chutes. Secondary reasons are: It is the safest method of mining; is suitable for steep veins; as the vein is composed of alternate bands of ore and waste with a large percentage of waste, there is an excellent

chance for sorting in stopes; the main working adit is of great length and is driven on the vein. Conditions were such that the ore was required immediately, and as there is a moderate pressure on the walls and filling will resist pressure too great for timbers, the levels can be kept open indefinitely by replacing rotten timbers. Narrow places in the vein do not cause so much dilution of ore as shrinkage stopes.

The disadvantages of the method are: A large percentage, usually from 12 to 25 per cent., of the valuable minerals are lost in the waste fill, as the rich fines enter the voids and the muckers fail to find the true floor. These losses could be reduced by shooting on plats. Obtaining and handling filling in places where the vein is wide, and the waste broken does not fill the void space, is costly. It is difficult to keep mill holes open, especially if the vein is wet and talc and gouge are present. An occasional soft rock in the wall of the mill hole will cut out, or a rock may be carelessly placed, and the mill may be lost. In places, especially where stopes come together and there is a pillar of waste, it is desirable to break the waste rather than leave the pillar; this makes excessive waste, which must be moved with a wheelbarrow or mucked to a mill hole. The total costs are greater than by the shrinkage system.

Shrinkage Stope

In the shrinkage method, highly inclined holes are used, breaking from a breast, or an opening in the back, if available. About one-third of the ore must be removed from the stope; the remaining ore gives a working floor for the miners and forms temporary support to the walls. When all of the ore in the stope is broken the remaining two-thirds of the ore may be removed. Greatest efficiency is acquired when the roofs of stopes are kept flat.

The flat vein of the Smuggler-Union Mining Co. is mined by a shrinkage method similar to that used at the Humboldt mine and by a modified shrinkage method. The dip of the vein is 59°, the width 36 in. The vein consists of alternate bands of ore and waste. The levels are 125 ft. apart. Chutes are 20 ft. apart. The manways are two to a stope, or 150 to 200 ft. apart on long stopes. Manways are stulled and lagged. The walls are fair. The drift on the vein is back-stoped, stulled or square-setted, then lagged. Portions of the vein below pay grade are left as pillars; also, pillars are left in places to support the walls. In the modified shrinkage system, the stope is carried up past a predetermined point for a level, then one or two rows of stulls are placed below the level (in most places one row is sufficient), lagged over, and 6 to 8 ft. of waste thrown on top of the lagging to bring the level to the required height. Openings are left next to the hanging wall, or 20 ft. apart, if the chutes are on 20-ft. centers. If the stope is too narrow

where the drift comes, the foot wall is shot out. The stope above is started in the usual manner and the chutes are spaced between the holes in the floor. When this work is completed, the lower stope is drawn empty and the upper stope started. Vein material is drawn on to a grizzly over the car and the waste is thrown directly into the open stope below. This method was adopted in order to obtain the advantages of the shrinkage system without diluting the ore with a large

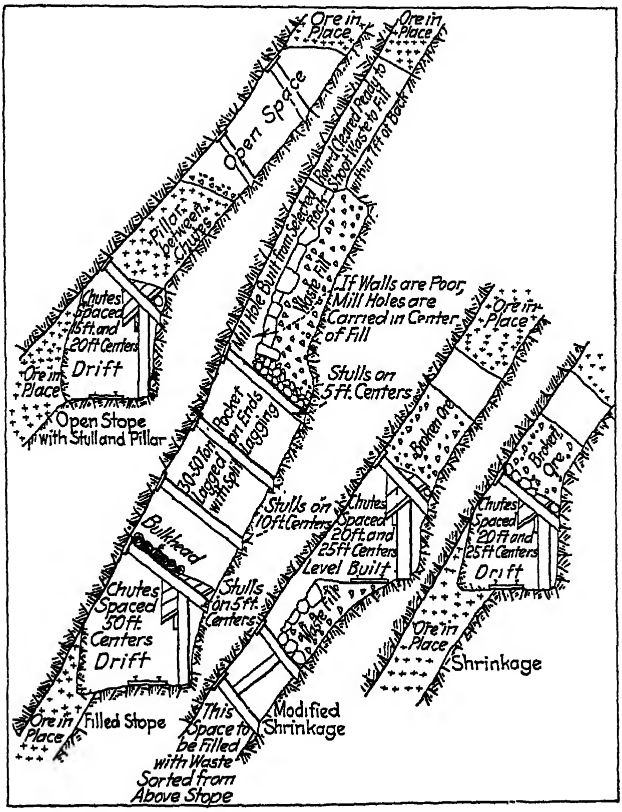

FIG. 1.

quantity of waste; also, to obtain the advantage of sorting before the ore is covered with muck, and to avoid the high tramming cost on the waste.

Built levels should be avoided where the walls are weak.

The Humboldt vein has a dip of 59° to 79° and is 14 to 24 in. wide. It is composed of alternate bands of ore and waste. The walls are strong. The distance between levels is 110 ft., as this was the original development; these levels, however, could be driven at 135-ft. intervals. A shrinkage system is used. The drift on the vein is back-stoped and lagged. The chutes are built on 20-ft. centers, and manways are placed

two to a stope, or 200 ft. apart on long stopes. All of the ore is trammed to two main ore passes, which are equipped with sorting stations; the waste is trammed to open stopes. This method gives the advantages of the shrinkage system and a close sorting of the waste.

At the sorting station, a chute is placed in a raise, which is to be used for an ore pass, about 40 ft. above the level. This feeds ore to an apex formed by two grizzlies, the long way of the bars being parallel to the length of the vein. Ore is deflected two ways, by the grizzlies, on to sorting tables at the bottom of the grizzlies, the fines dropping through the grizzlies into the main ore pass. The coarse ore is thrown into a chute at the side of the sorter and joins the fines, while the waste is thrown into a chute leading to the level below, from which it is drawn and trammed into an open stope for fill. A manway leads into the station from the level. The station is well lighted by nitrogen-filled lamps. Two, four, or five men work at these stations, depending on the tonnage passing through.

The cost of sorting ore in stations is 21 to 25 cents per ton, assuming that 80,000 tons go through any given station. The costs are divided as follows:

	Per Ton Material	Per Ton Ore
First cost station and waste track........	0.05	0.06
Tramming waste from station to stope...................	0.03	0.037
Sorting at 40 tons per man shift......	0.13	0.157
	0.21	0.25

Filled-stope sorting, that is, sorting the ore on top of the waste fill, mucking the ore to mill holes and the construction of the mill holes, had cost from 68 cents to $1.10 per ton of ore produced; direct sorting the ore as broken in stopes, then drawn on to a grizzly over a car, the waste thrown directly into open stopes below, cost from 6 to 10 cents per ton of ore produced and 28 cents per ton of waste produced. The cleanest product is made when sorting in waste-fill stopes, because the sorting stations and sorting direct allows all of the fines to go as ore.

The old stopes mined from the above properties are the result of the waste-fill method first described, where the fines were lost in the fill. Drifts are opened by spiling through the old fills, which were mined by the shrinkage method and sorted at the surface. When this work is resumed most of the waste will be left in the mine.

The old waste-filled stopes have been pulled with profit on company account in areas where the original ore was of good grade, but even lessees have had doubtful success in areas where the virgin ore has been of a moderate grade, *i.e.* $7 to $12 per ton.

The Tomboy and Black Bear mines are adjacent and connected and for many years were thought to be the same vein. While this is not true, the veins are similar and cross at a slight angle. The mining methods used are practically the same. The dip of the Tomboy vein is 80°, and the levels are 75 to 150 ft. apart. The dip of the Black Bear vein is 60°, and the levels are 140 ft. apart. Chutes are on 25 and 20-ft. centers; the walls are generally good, though weak in places. The levels are back-stoped and lagged, also level pillars are left with short raises above the chutes, to connect with the stopes. Shrinkage methods are used and the waste is left as pillars. The Tomboy sorts the larger pieces of waste at the mine portal before crushing.

The reasons for adopting this method were: The fissures are well filled with ore; the percentage of waste is small; the dip is steep and the walls are good; there is no loss of fines; staging, ore passes and shoveling are avoided; little timber is required, so that the method may be used on narrow or wide veins. The ore is strong enough to stand in the back of the stope and the shrinkage method is the lowest-cost method in camp. The disadvantages of the method are: Consistently weak walls cannot be mined by shrinkage method; if the dip is less than 60° a free settling of the ore is hindered; fairly continuous orebodies are essential, because stoping must be carried well in advance of the output; spalling of waste from walls reduces values; ore is lost, if the walls crush; ore left in stopes ties up capital; and the method is not flexible when started and difficult to change.

The main vein of the Liberty Bell Co. has a dip of 57° with a width of 3 or 4 ft. Levels are driven 125 ft. apart; walls are fair and poor. The drifts are stulled and lagged and chutes are built every 20 ft. A short raise is made at each chute; then a 45° incline is driven to connect with the incline from the adjacent chute, thereby leaving a pillar between the chutes. This pillar crushes by the time the stope is completed. Manways were placed at 75 to 100-ft. intervals. The spacing is variable; usually, there are three manways to a 250-ft. block. The stopes are open and stulled. This method was adopted because the hanging wall is weak and it is necessary to stull to avoid dilution of ore; a good support is given bad ground close to working faces and all of the ore broken is immediately available. The method has practically all of the advantages of the shrinkage method; its disadvantages are the cost of timber and the necessity for men to work on staging.

Another vein mined by the Liberty Bell Co. consisted of a high-grade streak 2 to 16 in. wide, sometimes in one and sometimes in three streaks, having the same value per square foot of ore regardless of the width. Numerous holes were placed in the ore but as few holes as possible were placed in the waste, the object being to break the ore fine and the waste coarse. This ore was mined by the shrinkage method, thence dropped through a sorting raise, and the waste used for stope fill.

The principal feature at the Liberty Bell is the retreating system, mining inward from the lateral extremes, and from the top down. This system concentrates a large part of the work on one level and leads to effective supervision.

Mine Openings

The drifts are driven 7 ft. high by 6 ft. wide, or the width of the vein if it is wider; one property carried the drifts 9 ft. high. Occasionally a contractor will carry the drift and back-stope as one breast. Crosscut tunnels are about 7 by 7 ft. and 7 by 5½ ft. Raises vary from 6 ft. to 17 ft. long by the width of the vein, unless the vein is very narrow. Shafts vary from 9 by 4½ ft. to 17 by 5 ft.

Drilling and Blasting

The latest types of Waugh, Ingersoll-Rand, and Sullivan drifting machines, stopers, and Jackhamers are used. Cleveland hitch cutters

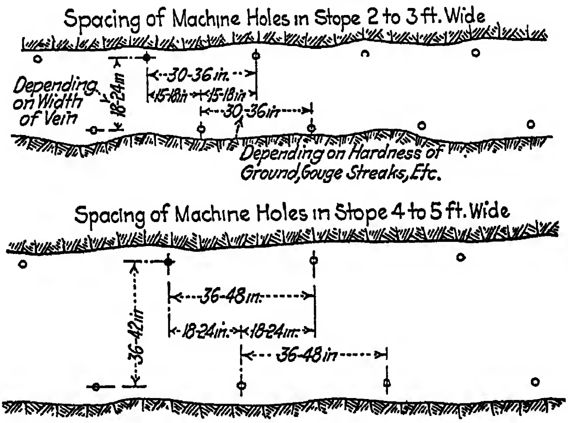

FIG. 2.

are used at two properties. For drifting 1⅛-in. and 1¼-in., round, hollow steel are used. The 5°–14° double-taper crossbit is used, though the high center crossbit is used for stoping in some of the mines. For stopers, ⅞-in. quarter-octagon, solid steel and 1-in. cruciform are used and for Jackhamers, ⅞-in. hollow quarter-octagon and 1-in. hollow hexagon with six-point rosette bits. The arrangement of holes in the drifts is not standard, because of the varying conditions of the veins. In general, the stopes are mined by the use of highly inclined holes 30 to 48 in. apart, the rows being 18 to 42 in. apart and arranged so that holes of adjacent rows are staggered. This arrangement is varied with the character of the ground. The loading of the holes, as well as the spacing, is important in order to avoid large pieces in the stopes and the subsequent breaking of these large pieces in the stopes and chutes; see Fig. 2.

During the war, ammonia explosives were tried but were not as satisfactory as the gelatin dynamites. The explosive would run out of a broken cartridge, and the cartridges did not tamp as well. With 30 per cent. dynamite the ore is broken in too large chunks; 40 per cent. is cheaper for general use. In very hard ground, such as the rhodonite in the Black Bear mine, 60 per cent. is necessary. In some of the drifting work, the contractors use a stick or two of 60 per cent. in the bottom of the holes. Tamping with clay has been tried, but the cost of obtaining and preparing the material is so great that it is as cheap to tamp with the dynamite. When using 40 and 60 per cent. dynamite together the caps are always placed in the 40 per cent., the primer being the stick next to the outside one.

Timbering

Round timbers are used exclusively, except in shafts. Creosote is sometimes used where timbered openings are main haulageways. Timbers are framed on the surface and at timber stations underground, then delivered to the working places by tramming crews. No timber is recovered for reuse. The timber slides from the levels into the stopes consist of two 2 by 12-in. planks for the bottom, two 2 by 6-in. for the sides. In the Liberty Bell open stopes, connections are made from the working stopes to the level above and timber is distributed to the stopes from the level above by means of small electric hoists. In other properties, timber is hoisted from the level below into the stopes with Little Tugger and electric hoists.

Tramming and Haulage

Loading is done from chutes; loading from development drifts is done by muckers. The work is done by 3, 4, and 6-ton, 250-volt, d.c., trolley locomotives, 4-ton storage-battery locomotives, and 3½-ton, 440-volt, a.c. locomotives.

Main haulage cars vary from 1 to 3¼ tons and are side dump, center dump, solid box, end dump, and gable bottom double side dump. Sublevel cars are of 1, 1½, and 2-ton capacity. The larger size are side dump and the smaller size end dump. Cars are dumped by hand, rotary dumps and compressed-air pistons.

One property has 24-in. gage track, all others use 18-in. For main haulage, 25 and 30-lb. are generally used, though 16-lb. rails are used. On intermediate levels, a 12-lb. rail is used. The grades average 0.5 per cent. in favor of the load.

Compressed-air main lines are usually 5 in.; lines on levels are 3 in.; lines into stopes 2 in. Ventilating pipes vary from 8 to 16 inches.

Continuous pillars are left above main haulage drifts, except when the ore is badly needed at the time the drifts are driven.

Power

The Utah Power & Light Co. offers two schedules as follows:

RATE No. P3. COMBINATION 24 HR.

First 100-hp. customers demand at, per month	$3.25
Next 400-hp. customers demand at, per month	2.25
Next 500-hp. customers demand at, per month	1.75
All additional hp. customers demand at, per month	1.00
In addition, for energy used: First 40,000 kw.-hr. of monthly consumption at, per kw.-hr	0.013
All additional	0.005
No discount,	

RATE No. P4. FLAT 24 HR.

First 100 hp. per month	$6.50
All additional	4.50

On account of the great distance underground, power is sent to underground transformer stations in armored, steel, submarine cables or lead-covered cables at 10,000 and 11,000 volts.

Hoisting

Most of the ore is dumped through long raises to main haulage levels. In the Liberty Bell mine, the ore was lowered in bottom-dump tubular skips. Hoisting is in balance. Hoisting ropes vary from $\frac{3}{4}$ to $\frac{7}{8}$-in. either plow steel or a better grade rope consisting of 6 strands and 19 wires, regular lay. Tugger hoists and small electric hoists use $\frac{1}{4}$ and $\frac{3}{8}$-in. rope. The Smuggler Union hoist is a double-drum type formed by placing a dividing flange in the center of a single drum. The dummy is a man skip, which is used only when men are going on or off shift. The main skip is the Liberty Bell type and is used for the transportation of men and supplies as the ore is passed through raises. The speed is 500 ft. per minute.

The hoist of the Colorado Superior Co. consists of a tramway grip sheave mounted vertically on a horizontal shaft. To avoid slippage, the rope is given additional contact with the grips by placing an idler pulley in front of the grip sheave. The rope from the under side of the grip sheave goes to a head-frame sheave, then to the skip and the rope from the top of the grip sheave goes to the idler mentioned, then to the head sheave, then to the dummy. The hoist is cheap to build and works satisfactorily, though a double-drum hoist with a clutch would be preferable if expense were no object. The speed is 500 ft. per min. The skip is self-dumping. The flanges of the rear wheels are wider than the front wheels, which allows the front of the skip to be depressed and the rear end elevated at the tipple. The three-point suspension principle is utilized for the wheels. This is an excellent scheme for an uneven track as all four wheels are always in contact with the track. All ore is hoisted.

The Humboldt hoist is similar to the Smuggler hoist, except that the dummy is cast iron. The hoisting equipment of Liberty Bell mine has been described by Charles A. Chase.[2] The ore was lowered with a skip as the main ore pass choked during the dry months, and the rest of the year enough water found its way into the chutes to give the soft ore the consistency of a thick soup. If the ore was allowed to accumulate and a chute broke, the trains would be buried. The novel features in this installation are the placing of the endless-rope mechanism for braking and driving with its drums in the plane of the shaft at the top of the shaft, thereby avoiding the deflection of the rope to and from it. The upper drum has two half laps of rope and the lower drum one half lap. The slippage is slight. A minimum amount of excavation is required for a hoist station. The skip is counterbalanced so that it is necessary to apply power only when an unusually large load is to be hoisted or the empty cage lowered. A 25-hp. motor operates the skip at 100 ft. per min.; while on gravity the speed is from 500 to 600 ft. per min. The skip is a tubular bottom dump with toggle lock having a capacity of 6 tons. The skeleton man cage, of great capacity for the light weight, was so successful here that the same type was used in several of the mines of the district. Safety catches are similar on all types of cages; they are cam-shaped dogs actuated by coiled springs on shafts.

Pumping

Pumping is mostly for water service. The pumps are 10-in. centrifugal, double and single suction and various sizes of small triplex pumps; all are driven by electricity.

Air Compressors

Ingersoll-Rand Imperial type 10 air compressors are most generally used, though there are a few Sullivan compressors. All are equipped with intake or intake and discharge type unloaders. The capacities vary from 200 to 1200 cu. ft. per min. Some properties use several small units and cut in units as the necessity demands. All are driven by electric motors.

Ventilation

Ventilation is natural in all but development headings, where blowers are necessary. Sturtevant and Buffalo exhausters, similar types of blowers, and Roots positive-pressure blowers, which will blow or exhaust by reversing direction of rotation are used. The blower is usually placed about midway the length of the ventilating pipe, giving equal distances for the intake and exhaust. There is little trouble maintaining satisfactory ventilation when there is any air circulation in the drifts.

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[2] Notes on Liberty Bell Mine. *Trans.* (1911) **42**, 694.

Main haulageways and stations are electric lighted. Individual miners use carbide lights.

Telephones and Signaling Systems

Complete telephone systems are maintained underground and on the surface. Two signal systems are maintained in shafts in addition to the telephone. One is an electric pull bell, which rings a bell and flashes lights at the hoist; the second is an emergency signal consisting of dry cells and two bare wires in the shaft, which can be short circuited with a knife or lamp at any point in the shaft.

Production

The tonnages produced by each of the larger mines has fluctuated from 150 to 600 tons a day. Stoping is done by day's pay and by contract. The stoping contracts are based on the fathom, which is an area 36 sq. ft. regardless of the width of the vein. The prices range from \$8 to \$12 a fathom for labor, carbide, and ammunition, depending on the character of the ground. Drifts are contracted by the foot of advance; the width called for is 6 ft. or the width of the vein, if wider. The contractor furnishes labor and ammunition. Prices range from \$7 to \$12 per ft. for this service.

Various methods of contracting work had been used. A former superintendent paid a bonus on the footage of holes drilled over 60 ft. per shift, but on account of the difficulty of measuring all the holes this was not entirely satisfactory. The system of contracting by the fathom works better, but it is difficult to prevent the men from breaking a stope wider than the vein.

Fathoms per machine shift vary from 0.87 to 1.21 in two properties. The Smuggler produces 1 ton per man per hour for miners in stopes, or 1 man-hour per ton. The Humboldt produces 1 ton per man per hour for miners in stopes, or 1 man-hour per ton. The Smuggler produces 0.34 ton per man per hour for all men underground, or 3 man-hours per ton. The Humboldt produces 0.27 ton per man per hour for all men underground, or 3.6 man-hours per ton. The Liberty Bell produces 0.313 ton per man per hour for all men underground, or 3.18 man-hours per ton.

H. G. McClain, former general superintendent of the Liberty Bell mine, does not have the tonnage per man in stopes and furnishes the following data: At \$4 per day for miners, the actual cost of mining ore was 30 cents per sq. ft. on vein widths of 5 ft. before the war. During the war, this rose to 80 cents, and was reduced to 50 cents by contracting during 1918–1919.

A fathom produces from 13.8 tons to 16 tons of ore to the mill after sorting out from 15 to 20 per cent. as waste.

As the surface labor includes freight rustlers, blacksmiths, drill repair men, aerial-tramway men, crushermen, car dumpers, watchmen, and office force, it would be confusing to include surface labor.

CLASSIFICATION OF LABOR FOR 9 MO. OF 1922, IN PERCENTAGE OF TOTAL BASED IN NUMBER OF MEN

	Smuggler Mine	Humboldt Mine
Shift bosses, foreman superintendents and office	6.4	4.9
Miners	33.5	27.1
Trammers	31.0	40.9
Ore sorters	2.8	10.0
Timbermen	20.9	9.1
Hoist and compressor men	4.6	7.5
Sundries	0.8	0.5

TOTAL COST, NOT INCLUDING DEVELOPMENT, IN PER CENT. OF TOTAL MINING COST

Tons		Labor	Supplies	Power
117,025*	Smuggler	60.9	33.2	5.9
63,493*	Humboldt	61.0	34.0	4.7
988,325	Liberty Bell	66.0	30.7	3.37[a]

[a] Total supplies 34 per cent.; power estimated to be same as in 1910 and deducted from 34 per cent. supplies, giving 30.7 for supplies and 3.3 per cent. for power.

* 11 months 1922.

EXPLOSIVES, POUNDS PER TON

	Shrinkage	Waste Fill	Open Stulled
Smuggler	1.33	1.47	
Liberty Bell[a]	1.31[b]		1.03
Humboldt	1.75	2.4	

[a] Vein wide and easy to break, as it contained considerable calcite.

[b] Vein hard, similar to Smuggler. About 15 to 20 per cent. waste was sorted from this, as in the Smuggler.

TIMBER CONSUMPTION AT SMUGGLER AND HUMBOLDT MINES

Tonnage for 2 years	389,887*
Lagging, 10 ft. long, number of sticks per ton	0.1368
Poles 6–9 in. at small end, linear feet per ton	0.2753
Stulls 9 in. and larger at small end, linear feet per ton	0 2989

* 11 months 1922.

The timber adjacent to the mines has been cut so that mine timber must now be shipped from 20 to 30 miles. Sawn lumber is shipped in from New Mexico and Oregon.

Coal is shipped in from Hesperus and Bowie, Colo.

Ore Mined

Waste fill, 10 to 25 per cent. of ore is lost in fill. Open-stulled stope width was 3 ft., stope mined 4.3 ft. With a 4½-ft. vein, the dilution is 20 to 25 per cent., by comparing mill tonnage with assay tonnage. Other veins vary from 10 to 40 per cent. excess, depending on width of vein and character of walls.

Disposition of Ore after Reaching Surface

Dumped over grizzlies to crushers; then, at Smuggler, Black Bear, and Liberty Bell, the ore is sent over aerial tramways to the mills. At the Tomboy, the ore is sent direct to the mill.

Safety and Welfare Work

Safety work is carried on under the supervision of the state. There are first-aid stations at the mines and an excellent hospital in Telluride. Families of employees are furnished medical services for a small fee.

Schools are maintained by the county at the large mines.

The mines have clubhouses containing pool tables, card tables, picture shows, and general stores well stocked with every article a miner cares for.

DISCUSSION

Charles S. Hurter,* Wilmington, Del. (written discussion).—At the top of page 559, the statement is made that the cost of obtaining and preparing tamping material is so great that it is as cheap to tamp with dynamite. It would be of considerable interest if this part of the subject would be gone into in more detail. The Bureau of Mines, Technical Paper No. 17 entitled "The Effect of Stemming on the Efficiency of Explosives" shows that the gain in efficiency varied from 30 to 60 per cent. when the hole was tamped properly to the collar. The conditions under which these test were made probably never could be obtained in practice. They gave a gain of about 30 per cent. with the 40-per cent. straight nitroglycerin dynamite. Gelatin, when properly loaded in a bore hole, is about 10 per cent. quicker in its action than the corresponding grade of straight dynamite. On the basis of these figures, the writer believes that there should be a gain with 40-per cent. gelatin of about 20 per cent. by tamping the holes properly. The most noticeable gain is in the first foot or two of tamping—what the miners call "making the hole air-tight"—but there is still further gain, according to the Bureau of Mines, when the hole is tamped solid clear up to the collar.

It would be interesting to know on what basis the author figures that the cost of tamping was too great. A hole can do no more than pull its burden. Was the tamping used with the idea of increasing the amount of work done, or did they try to cut down the charges 20 per cent. and tamping the holes solidly to the collar? In addition, the fumes from a round of shots properly confined by good tamping are far less than the

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* Technical representative, Du Pont Co.

fumes of shots where no tamping at all is used. It is not necessary to use clay. A clayey loam or similar material made up into dummy cartridges is plenty good enough. Dry sand is all right for flat holes.

Charles N. Bell (author's reply to discussion).—There was no material suitable for tamping within several miles of the mine. It was necessary to haul the material to the mill, and two transfers of material were then required. A man was employed to fill the tamping bags, which were then packed in powder boxes for transportation over a wire-rope tramway and underground haulage. The long distances over which the bags were distributed kept a large percentage underground long enough for the paper bags to absorb moisture from the air and the tamping material, which caused a two-fold loss. The tamping material was a little more difficult to load in the holes than powder. When all factors were considered we found that there was little saving.

Shovel Operations at Bingham, Utah Copper Co.

By H. C. Goodrich,* Salt Lake City, Utah

(Salt Lake City Meeting, September, 1925)

At the Utah Copper mine, steam shovels were first used, in 1906, for the removal of overburden, and in June, 1907, for the mining of ore. Prior to 1907, the ore came from underground development work and from stoping. The use of the steam shovel gradually increased until 1914; since then all material, both overburden and ore, has been loaded by power shovels operating on the various benches of the mine. During

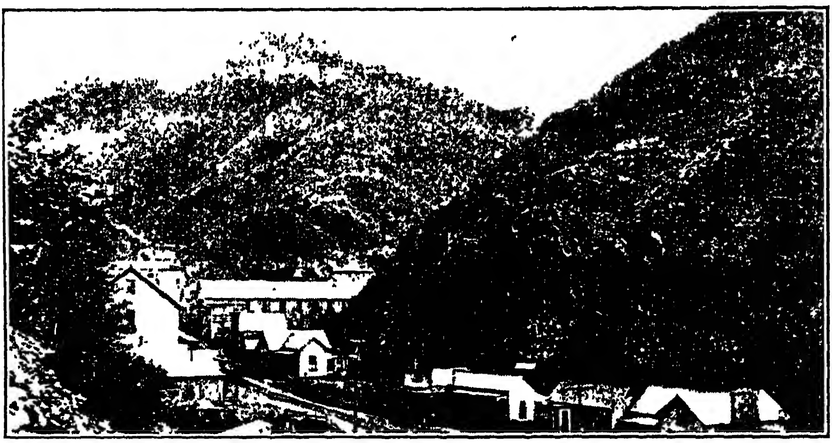

Fig. 1.—Scene of Utah copper mine before open-pit mining was begun.

these 19 years, there has been a great improvement in open-pit mining, largely in the use of equipment having greater capacity and dependability. The history of the development of mechanical shovels and related equipment, as affecting the Utah Copper Co., would be interesting and would justify considerable space; but of greater interest at this time is the operation of the equipment now in use.

The Utah Copper mine is in a great deposit of copper-bearing monzonite, which occurs as a prominent headland of the Oquirrh range of

* Chief Engineer, Utah Copper Co.

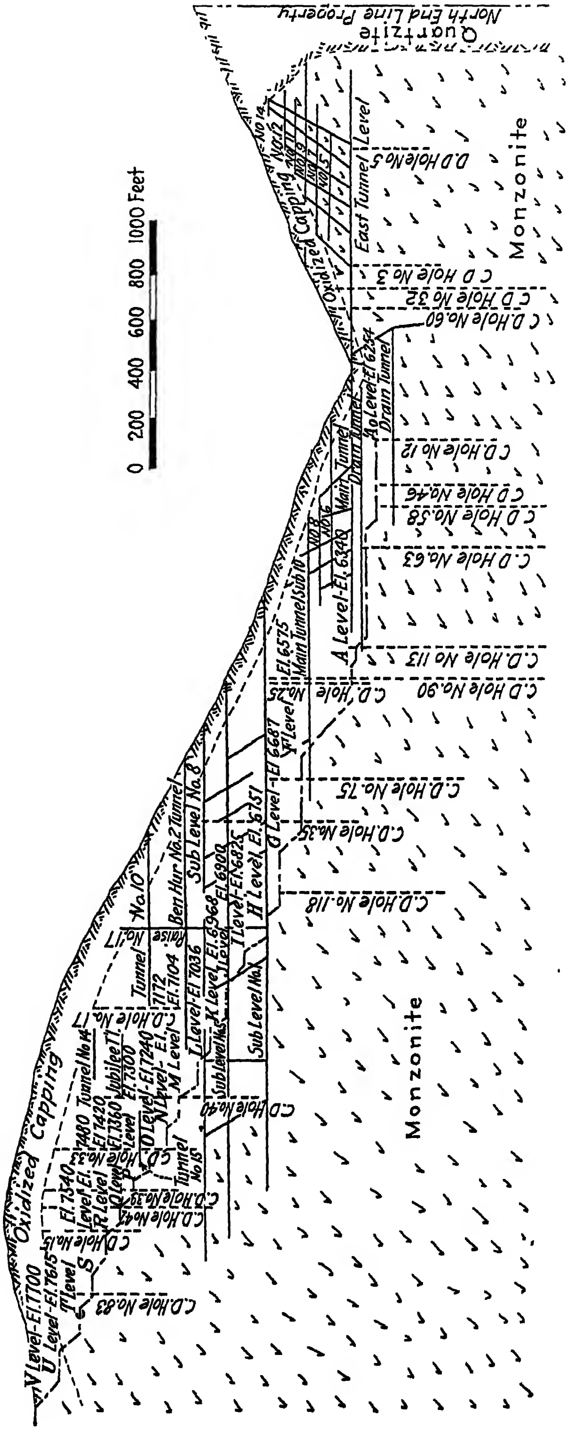

FIG. 2.—LONGITUDINAL SECTION THROUGH CENTRAL PART OF OREBODY, JAN. 1, 1925.

mountains overlooking the town of Bingham and about 22 miles southwest of Salt Lake City. This separate and distinct mountain, lying between main Bingham Canyon and Carr Fork, is shown in its original condition, in Fig. 1. In the early days, the Utah Copper Co. owned property at the bottom and the Boston Consolidated Mining Co. owned the property at its top. The former acquired the latter's holdings early in 1910, and immediately thereafter outlined the system of shovel benches that now form so prominent a feature of the topography. The opening for exploitation of the Utah Copper mine was done in part by the company's own steam shovels and, in part, such as building bridges and excavating for railroad tracks, under contract.

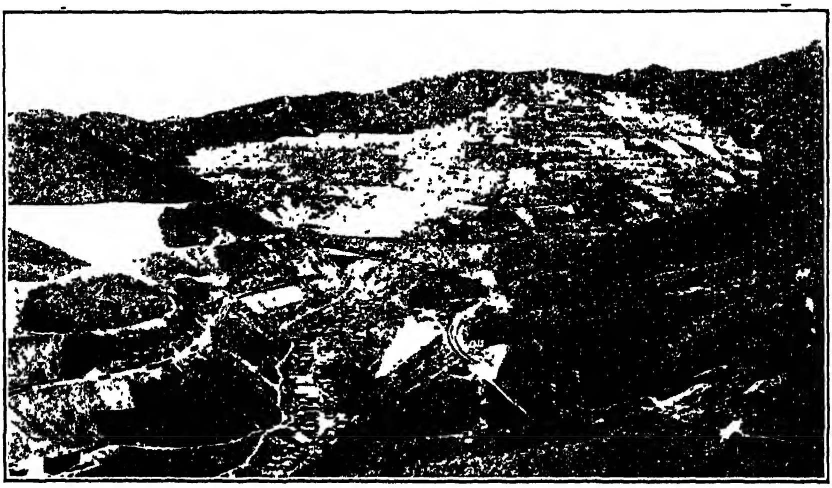

Fig. 3.—Switchbacks and dumps in right background, pit in center and main Canyon switchbacks and dumps on left; part of Bingham & Garfield Ry. in foreground.

The early investigations showed that overlying the ore, and through the gulches, the surface material consisted of earth, loose rock, and some decomposed porphyry, and that in many places on the ridges solid porphyry rock was exposed, the exposed material being harder than the ore lying beneath. At that time it was decided that this should be an open-cut mine, all material to be handled by shovels, the stripping to be loaded into side-dump cars and hauled to and dumped into the nearby gulches, the ore to be loaded into railroad cars for transportation to the mill. These two operations are kept separate and distinct.

Both the surface material and the ore, which is a porphyritic rock, are intensely fractured and break well with primary blasting. This is true not only of the ore within the mass of the mountain, but also near its contact with quartzite where it has been metamorphosed into a harder rock having physical qualities greatly resembling those of quartzite.

A characteristic longitudinal section through the central portion of the orebody is shown in Fig. 2; and the operation of moving stripping and ore in Fig. 3. These figures show the operation of removing the stripping from the ore and the transporting of the ore down to the assembly yards at the base of the mountain where the trains are made up for movement to the mills. Operations are conducted every day of the year in spite of the fact that the winters, sometimes of 8 months' duration, are generally severe and have many days of high winds, heavy snows, and low temperature.

Equipment Used

There are 58 standard-gage steam locomotives handling the ore and stripping trains at the mine. The first ones weighed 50 tons each; but in 1916, the company purchased 75-ton locomotives, which can haul ten empty ore cars up the 4 per cent. grade. Originally the ore was loaded directly into 70-ton all-steel ore cars, of which there were 650; commencing in 1922, these cars were rebuilt to 80-ton capacity. The dump cars originally used for the stripping were of 6 cu. yd. capacity; but in 1910, 12 cu. yd. steel dump cars were substituted, and in 1916 the 30 cu. yd. all-steel air-dump cars were adopted as standard equipment.

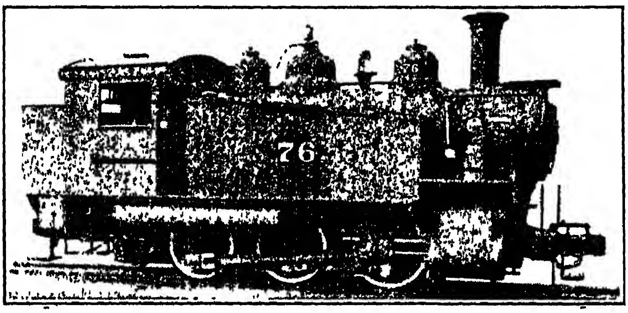

Fig. 4.—75-ton switching locomotive.

The shovel equipment has been added to from year to year so that now there are:

Five Marion Model 91 steam-type, caterpillar-traction, 3½-yd. dippers.

Six Marion Model 91 steam-type, caterpillar-traction, 4½-yd. dippers.

One Marion Model 91 steam-type, caterpillar-traction, 5-yd. dipper.

Two Bucyrus Model 95-C steam-type, caterpillar-traction, 4½-yd. dippers.

Three Marion Model 91, converted from steam to alternating-current caterpillar-traction, 4½-yd. dippers.

Five Marion Model 92, alternating-current, caterpillar-traction, 4½-yd. dippers.

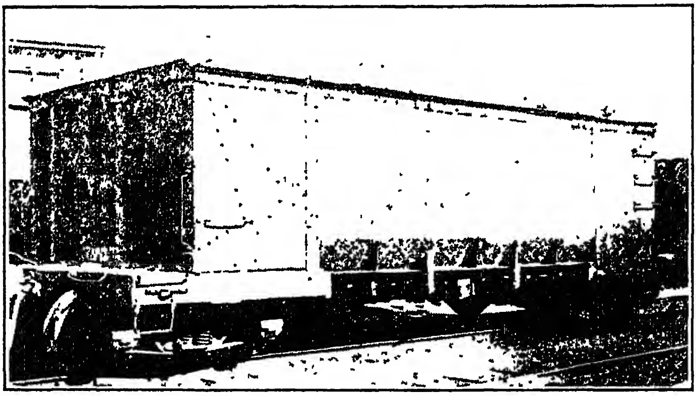

FIG. 5.—80-TON ORE CAR.

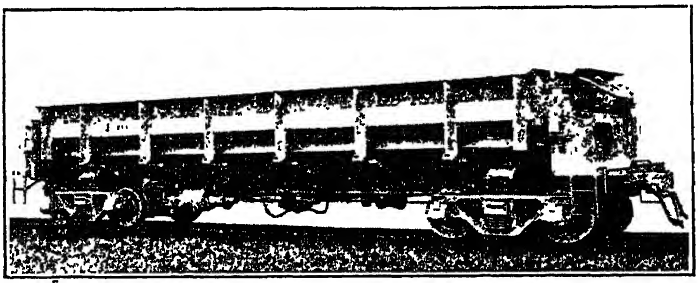

FIG. 6.—30-CU. YD. DUMP CAR.

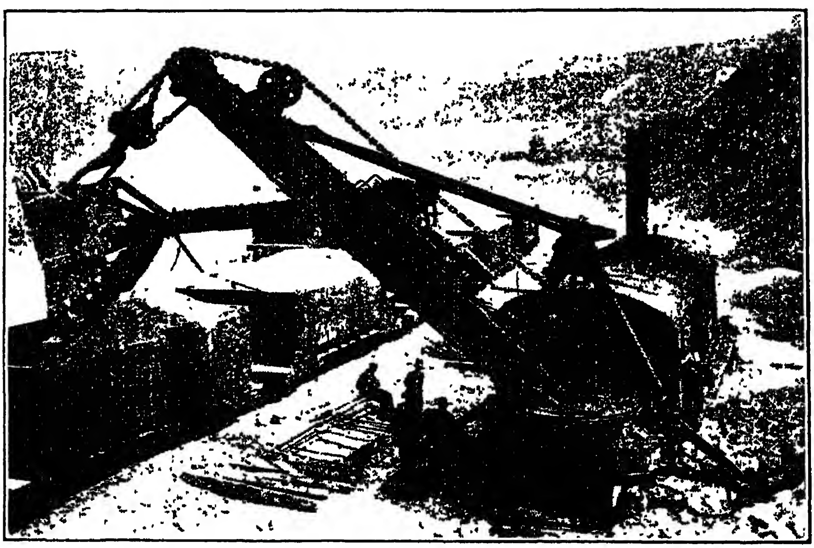

FIG. 7.—STEAM SHOVEL, RAILROAD TRACTION, LOADING 80-TON ORE CAR.

One Marion Model 92 direct-current, caterpillar-traction 4½-yd. dipper.

It will be noted that this equipment does not include steam shovels with 3½-yd. dippers mounted on railroad traction, although they are

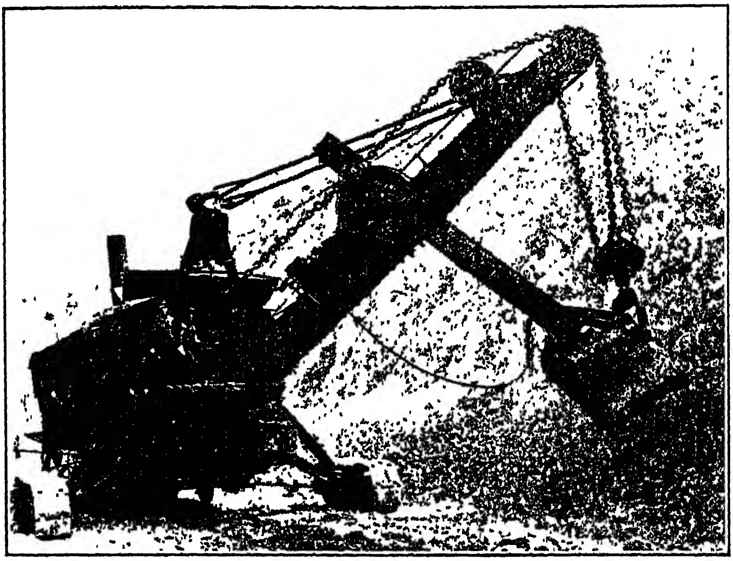

FIG. 8.—STEAM SHOVEL, CATERPILLAR TRACTION.

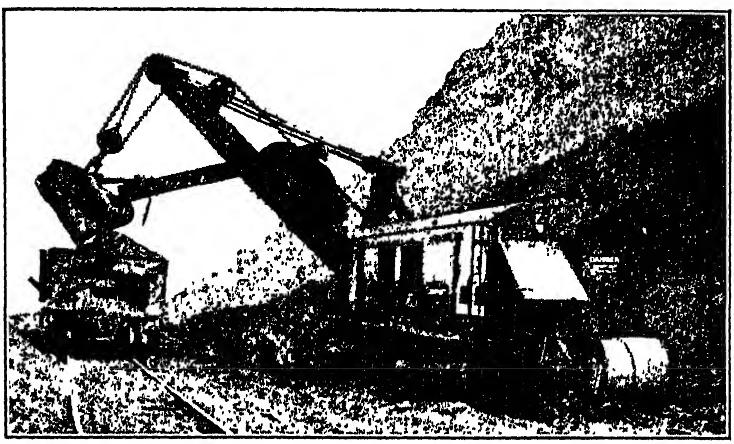

FIG. 9.—ELECTRIC SHOVEL USING DIRECT CURRENT.

mentioned; the foregoing is the equipment now in use but the paper gives shovel performances for a period of 16 months. The types not shown in the lists, but mentioned, have been abandoned. The latest equipment at the Utah Copper mine is the 100-ton railroad-type, electric shovel,

either so-built, or converted from a steam shovel by substituting electricity for steam, on caterpillar tractors, and having a 4½-cu. yd. dipper.

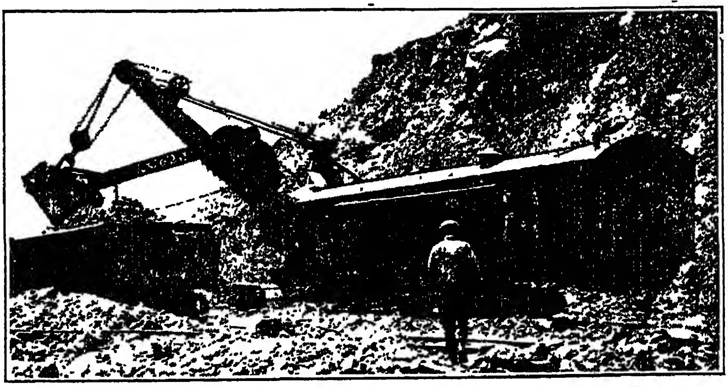

FIG. 10.—ELECTRIC SHOVEL USING ALTERNATING CURRENT.

In order that shovels loading into cars may operate economically they must be continuously supplied with empty cars. Early in 1910, because the transportation service was inadequate and did not keep the shovels supplied with empty cars, resulting in a loss in capacity of the

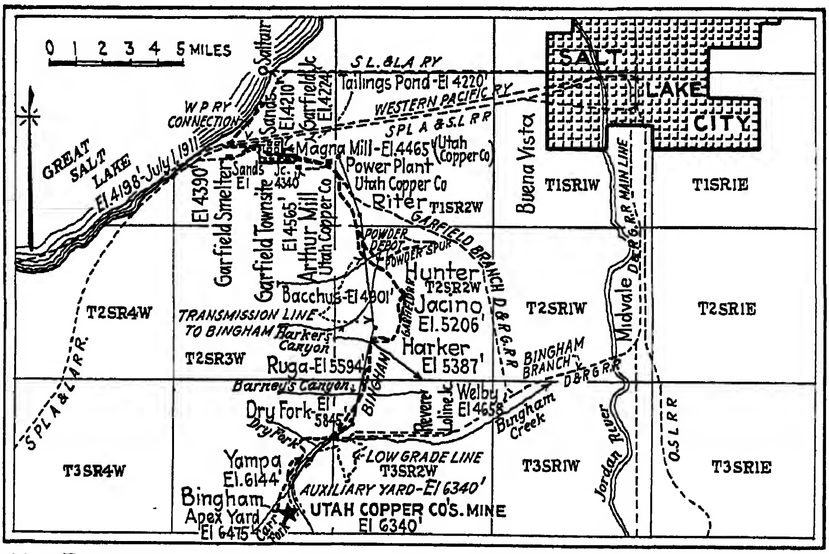

FIG. 11.—PORTION OF SALT LAKE COUNTY, SHOWING RELATIVE PORTIONS OF UTAH COPPER CO.S' MINES AND MILLS.

shovels, the Utah Copper Co. started the construction of the Bingham & Garfield Ry. to connect its mine with its mills; this line was completed and put in operation on Sept. 14, 1911. The location of this railroad is shown in Fig. 11.

A switchback system of tracks at the mine, laid with 65-lb. rail, with maximum curvature of 16° and maximum gradient of 4 per cent. was constructed under contract after the completion of this railroad and the top level, elevation 7760 (stripped off in 1914) was reached and opened up during 1913. Before the construction of the Bingham & Garfield Ry., when the mine was using 50-ton locomotives and 50-ton ore cars, the 65-lb. rail was sufficient to carry this traffic; but when the 75-ton locomotives and 80-ton ore cars were introduced it was necessary to replace the 65-lb. rail with the 90-lb. rail up to the I level. There are dump tracks from the different benches to the nearby gulches for the disposal of stripping. Where it is necessary to haul waste material long distances, the stripping disposal lines are located at such elevations as to serve two or more benches. There are approximately 50 miles of standard-gage mine track.

The water available in the Bingham Canyon district was of poor quality for culinary purposes and steam boilers. So, in 1913, pipelines were laid to bring in an ample supply from Tooele County. This pipeline passes through 2¼ miles of tunnel, which cuts through the divide between Middle Canyon, the source of the water, and Carr Fork Canyon.

Organization of Forces and Work

The organization of the forces at the mine is shown on the organization chart, Fig. 12; this chart is self-explanatory.

The average number of men employed was 1873. The classification of the employees, the number of men, and the man-shifts worked, averaged for winter and summer months, is as follows:

Number	Classification	Man-shifts Worked
110	Supervision	3,153.8
193	Drilling and blasting	6,053.0
234	Shovel operations	6,882.6
391	Train operations	11,605.7
543	Track operations	16,667.3
449	Mechanical department	13,016.4
22	Miscellaneous	687.2
1942		58,066.0

On an average of 1,240,230 cu. yd. handled per month there were:

205 cu. yd. per man-shift, drilling and blasting
180 cu. yd. per man-shift, shovel operations
107 cu. yd. per man-shift, train operations
74 cu. yd. per man-shift, track operations
21.4 cu. yd. per man-shift, for all employees

In 1910 (steam-shovel operations at the Boston Consolidated mine having been discontinued) the Utah Copper Co. had the shovel benches

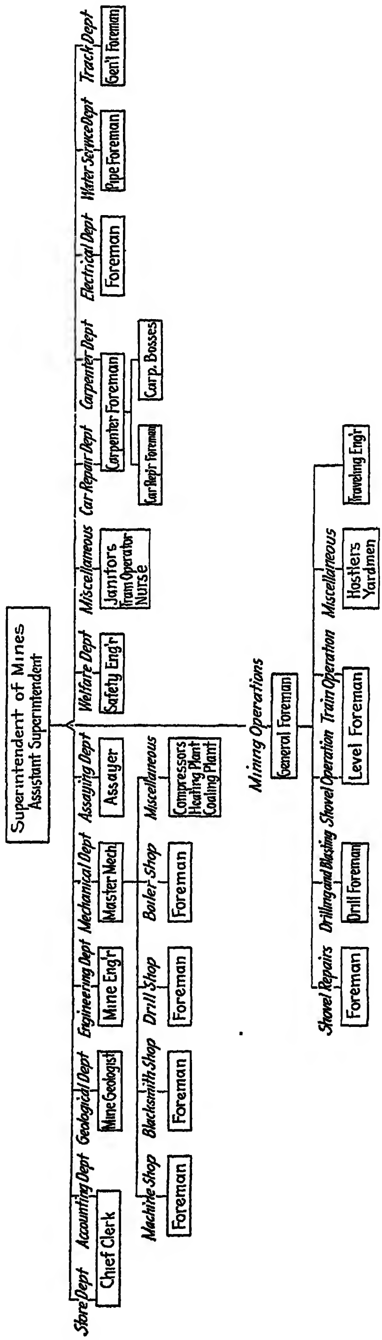

FIG. 12.—ORGANIZATION CHART.

opened up from A level, at the bottom of the mountain to J level, about half way up, and stripping lines C, E and H constructed on the east side of the canyon. The problem before the engineers at that time was the establishment of the benches from J level to the Boston Consolidated workings, the reestablishment of benches used by that company to the top of the mountain and the connection thereof, by railroad lines, with the assembly yards at the bottom. As experience had shown that railroad cars loaded with ore could be handled by the 50-ton mine locomotives on 4 per cent. grades, with safety and dispatch, and that these locomotives could haul five empty cars up the hill, this grade was adopted as maximum for the switchback tracks. Tail tracks of length sufficient to pass these trains were built at the end of the switchbacks; these were extended to accommodate trains of ten cars when the 75-ton locomotives were acquired.

There are now twenty benches, which are substantially level and traverse the entire face of the mountain. The vertical distance between them averages 70 ft., the maximum being 225 ft. and the minimum 46 ft. The lowest bench is at elevation 6240 ft. and the top bench at 7700 ft. The average length of the benches is approximately 1800 ft.

Drilling and Blasting

The drilling and blasting of the benches is done by crews, of which there are two to each shovel. A crew consists of a machine man, a helper, and a bank man. One crew follows behind the shovel and drills the toe holes, while the other works ahead of the shovel and on the muck pile, drilling what are locally known as bank holes. The toe holes, drilled at an angle of 5° to 15° from the horizontal, are spaced from 10 to 15 ft. apart and are about 20 ft. in depth. There is no regularity in the spacing of the bank holes; they are drilled at places in the bank where the blasting of the toe holes did not break the bank to a normal slope but left masses of rock projecting from the face. The drills used are air drills mounted on tripods. The holes are started with a 3.75-in. gage bit and are tapered to 2.25 in. at the bottom of the hole. An average of 46 ft. is drilled per shift.

After being drilled, the holes are chambered from four to seven times, until they are sufficiently large to receive the required charge of powder, which averages 185 lb. to the hole. If the hole is ragged, the powder is introduced through a 1½-in. pipe which is removed before blasting. After the hole is loaded, it is blasted by using a priming charge, which consists of a stick of powder with a cap and a 6.5-ft. length of fuse. The latter is ignited and the primer inserted in the hole with a tamping stick about 25 ft. long.

Ordinarily the blasting is done during the lunch hour or at the end of the shift, although there is no regular time for blasting the bank holes;

they are fired whenever the shovel has dug out the pit. Precautions are taken when blasting to prevent accidents. Prior to setting off any blasts, the whistle on the shovel nearest the blast is blown as a warning; in addition an air siren in the lower yard is sounded; this siren can be heard for a long distance.

The trimming of banks is done by the drilling and blasting crews. The spacing and pointing of the holes and the charge of powder used are determined by the powder foreman. Experience has shown that this results in best breaking of the ground to such a size that it can be easily handled by the shovel. Boulders too large to go through the dipper are broken either by bulldozing or by drilling short holes into them with a Jackhamer. The powder is a 60 per cent. ammonia non-freezing powder manufactured at a plant located on the Bingham & Garfield Ry. about 10 miles from the mine. An average of 3.94 cu. yd. of material is broken down per pound of powder used.

The air to supply the drills is piped to each bench by a continuous air line, which goes completely around the workings and is supplied from the main compressor plant, which has a capacity of about 8000 cu. ft. of free air per minute and is centrally located. Fittings are placed to facilitate connection to this air pipe at both ends of each bench for use thereon.

Operation of Shovels

Shovels located on the upper benches are coaled from standard railroad cars (as they come from the coal mine) placed in position by tramp locomotives during the night. A man at each steam shovel unloads the coal from the car on to a shoveling sheet and from this into the shovel coal box. The shovels on the lower benches are supplied with coal delivered in 12-yd. steel dump cars, which are filled at a modern type of railroad coal pocket, illustrated by Fig. 13, situated at the auxiliary yard and hauled from there to the various benches by tramp locomotives. Locomotives on duty with shovels on the Main Canyon side get their coal from this coal pocket. Locomotives serving ore shovels and using the switchbacks on the Carr Fork side obtain their coal from the same car as the shovel; there would be too much delay if these locomotives were to set out loads and go to the auxiliary yard, as the end of their haul is in the Apex yard, about a mile from the coal pocket.

One of the two locomotives serving each shovel in ore is tied up with the shovel during the night, in order that the shovel may start loading immediately on the beginning of the shift; the other locomotive is tied up in the auxiliary yard, preparatory to taking the first string of cars up the hill in the morning. These locomotives are coaled, sanded, and watered and their fires cleaned at the coal pocket during the

night. Where the locomotives coal at the shovels, there is an engine coal man. All locomotives engaged in hauling waste are also coaled at the shovels.

On each bench where steam shovels are working there is a 2-in. water line, liberally supplied with unions. When watering the shovel, a short length of 2-in. rubber hose is attached to the end of the pipe and connected with the tank on the shovel. When the shovel is moved ahead, a length of pipe is removed and the hose reattached. Locomotives are watered from water tanks conveniently located.

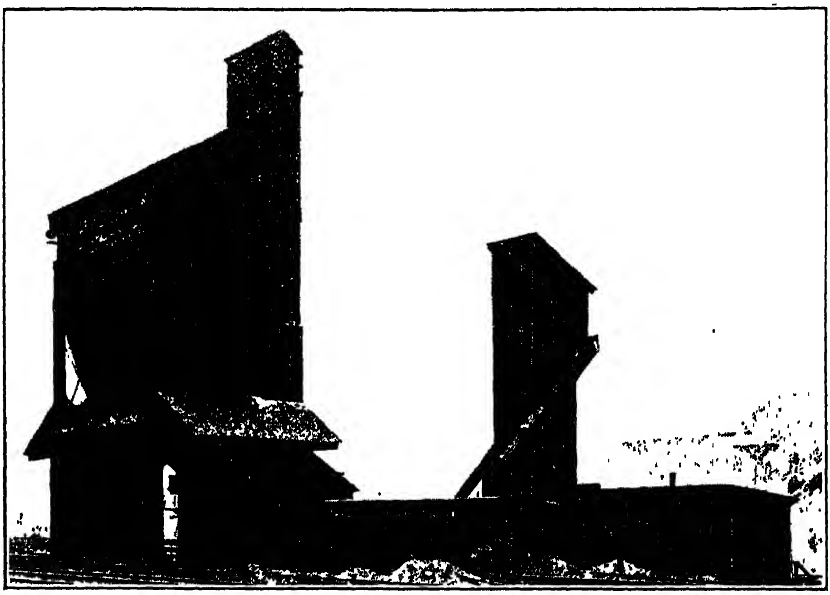

Fig. 13.—Coal, ash, and sand station.

Each morning, the mine is informed by the mills of the ore requirement for that day. It is the policy to work as many shovels during the 8-hr. daylight shift (7:00 a.m. to 3:30 p.m.) as is possible. In describing a shift of shovel work it is necessary to go back to the preceding shift, for it is at the end of that shift, or shortly afterwards, that provision is made for the commencement of operations on the succeeding shift.

Each shovel is served by two locomotives; at the end of the shift one of these is tied up with the shovel with or without empty cars, depending on whether empty cars were available in the assembly yard when the locomotive made the final trip up the hill. If empty ore cars are available 15 min. before quitting time, the locomotive takes up sufficient empties to keep the shovel busy until the other locomotive can get to it with cars

after the beginning of the shift. Shovels that do not have sufficient empty cars at the end of the shift are supplied by a tramp locomotive working between shifts, so that regardless of the number of empty cars available at the end of the shift there are cars at the shovel, at the beginning of the shift, for it to start loading.

With two exceptions all of the switchbacks are single track, so the movement of trains on them is controlled by flagmen placed at advantageous points. During cloudy or snowy weather, when it is impossible to use visual signals, there is a telephone system used exclusively for this purpose. Whenever a train is ready to leave a bench, either upward or

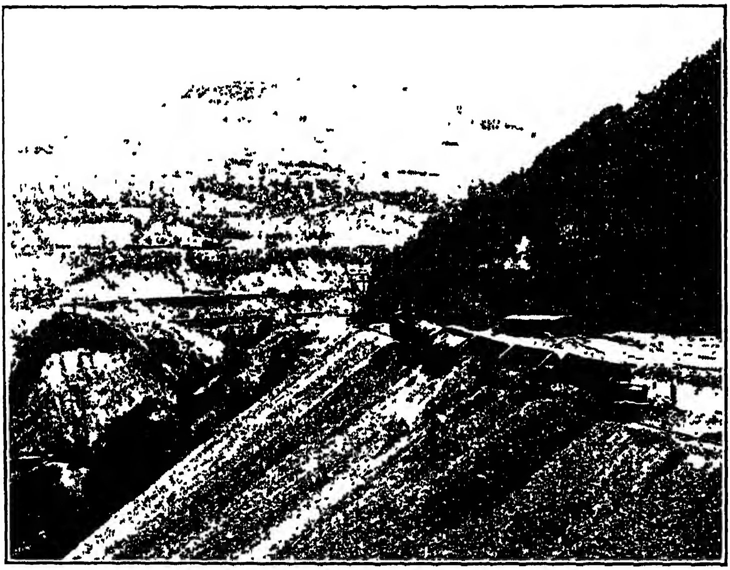

Fig. 14.—Dump train unloading; Carr Fork switchbacks in background.

downward, the engineer asks for a signal by a whistle; if given permission to leave by the flagman in charge of that section, he proceeds to the division controlled by the next flagman, where he again has to whistle for signals.

Shovels in stripping are served by two locomotives, as are the shovels in ore. The dump lines from the higher benches are continuations of the bench lines into the adjacent gulches. The lower group of benches is served by dumps common to two or more benches. This involves the handling of waste trains on the same switchback tracks as the ore is handled. The 50-ton locomotives handle the waste trains on the top level dump tracks, where the haul is not very long; on the lower benches, where the haul is longer, the 75-ton locomotives are used, with trains of five or six 30-yd. dump cars. When loaded cars are taken down from a bench above the common dump line, the trains are the same as when

hauling on the common-bench dump line, but when the loaded cars must be pushed from a lower bench to a common dump line, an extra locomotive is required to help the train up the switchbacks.

In disposing of stripping, the first dumping is at the end farthest from the shovel bench. As this part of the dump is filled, the material is dumped successively closer to the shovel bench, until the entire length of the dump for that position of the track has been filled. When the dumps have been filled to the outer end, the dump gang levels off the dump and shifts the track to the edge. One dump gang, consisting of a boss and approximately twenty men, takes care of three dumps where each dump is serving but one shovel. Where the trains from two or more shovels run on to one dump and dumps are high the entire services of one gang are required for that one dump.

In their travel across the benches the shovels are not permitted to operate directly above one another, but are moved in echelon. After a shovel has passed to the extreme end of a bench, a track gang, which is the consolidation of from three to five of the smaller gangs of 20 men, shift the track into position for the next cut through. Track shifting is done with lining bars.

Samples are taken at 10-ft. intervals behind the shovels. The results of the assays are placed upon a vertical projection and are referenced to coördinate lines thereon. The positions of these lines, as they cross the various levels, are located on the track ties by metal markers. Thus the grade of the ore is always known at positions easily located by those responsible for the change from the loading of ore to waste or vice versa at the proper time. While it is the policy to operate as much as possible in the day shift, some work is done at night and this night work is conducted in such a way as to preserve a proper balance in the working of the mine.

At the end of the day's work, a check is made of the amount of ore that has been loaded and enough shovels are detailed to make up the deficiency on the night shift. As the grade of ore loaded on the day shift is known, within reasonably close limits, the location of the shovels that are to work at night depends partly on the quality of ore necessary to be mined to maintain an average grade. The disposal of the shovels on the night shift is also made so as to preserve the uniform progress of the shovels across the benches. As all shovels have not the same capacities some will travel faster than others across the benches, so the night shift is utilized to maintain the correct relative positions as nearly as possible.

In addition, the night work aids in keeping the progress of the various dumps uniform and to preserve the dumping space. It is necessary to maintain the advance of dumps situated one above the other in order that trains dumping on the different benches will not interfere with one another. Such variations can be corrected by stripping at night on the benches whose dump is lagging.

The employees are transported to and from work in specially designed coaches, which stop on each bench during each trip.

Maintenance of Equipment

The boilers of steam shovels are washed out once a month and repairs to the shovel are made at that time. It is seldom necessary to take shovels to the machine shop for repairs, as it is generally possible to repair them on the benches. One steam shovel is kept steamed for reserve and for an emergency.

Four switching locomotives are washed out each day, three at the lower shop and one at the Boston shop. Ordinarily two of the fifty-eight locomotives are undergoing complete overhauling in the back shop. There are two repair shops for the upkeep of dump-cars. There is one car repairman in the field for each three benches; he makes such light repairs as setting and adjusting brakes, packing journal boxes, etc.

Results of Operation

Table 1, which shows the yardage of ore and stripping handled by Utah Copper shovels from the summer of 1906 to the end of 1924, gives

Table 1

Year	Ore, Cubic Yards	Stripping, Cubic Yards	Total, Cubic Yards
1906		110,408	110,408
1907		627,631	627,631
1908	822,330	967,283	1,789,613
1909	1,181,790	1,526,470	2,708,260
1910	1,807,770	2,145,844	3,953,614
1911	1,759,410	3,410,621	5,170,031
1912	2,100,780	2,921,824	5,022,604
1913	3,531,000	3,742,993	7,273,993
1914	3,251,540	5,296,533	8,548,073
1915	4,276,632	5,961,367	10,237,999
1916	5,687,383	5,911,455	11,598,838
1917	6,073,274	4,143,201	10,216,475
1918	6,030,655	3,817,008	9,847,663
1919	2,684,407	1,964,261	4,648,668
1920	2,721,083	2,754,041	5,475,124
1921	611,421	355,231	966,652
1922	2,070,355	1,101,753	3,172,108
1923	5,382,063	2,517,025	7,899,088
1924	5,774,095	5,945,588	11,719,683
Total	55,765,988	55,220,537	110,986,525

a good idea of the magnitude of the operations and the work the shovels have done. This total quantity would form a prism 1¼ mile high on a Salt Lake City block (660 ft. square).

During March, 1925, the shovels handled 716,526 cu. yd. of stripping, which was hauled an average of 1.09 miles to the dumps; and 991,355 tons of ore, which was transported an average of 3.61 miles from the shovels to the assembly yards.

Table 2 shows the operation and performance of all types now in service for the 16 months from Jan. 1, 1924 to May 1, 1925.

TABLE 2

	Shifts		Output, Cubic Yards		Average Output, Cubic Yards per Shift	
	Ore	Waste	Ore	Waste	Ore	Waste
Steam shovel 3½-yd. dipper, railroad traction	325.61	548.50	275,994	494,414	847.62	901.30
Steam shovel 3½-yd. dipper, caterpillar traction	784.14	2,494.76	898,799	3,225,815	1,146.22	1,293.04
Steam shovel 4½-yd. dipper, caterpillar traction	3,351.77	1,502.51	4,774,049	2,120,016	1,424.34	1,410.98
All electric shovels with 4½-yd. dipper, caterpillar traction	650.52	1,021.75	977,987	1,322,150	1,503 39	1,294.01

Table 3 is a comparison of the non-operating time while on shift, expressed in percentage of the total of such time, for the same types and over the same period.

TABLE 3.—*Percentage of Non-operating Time While on Shift*

Causes	Steam Shovels			Electrics	
	R. R. Traction	Caterpillar Traction			
	3.5-yd. Dipper	3.5-yd. Dipper	4.5-yd. Dipper	A. C.	D. C.
Derailment of shovels	1.16				
Other, not common to all types	10.26	12.58	12.59	16.36	6.30
Account of locomotives	0.68	0.41	0.52	0.44	0.43
Shovel repairs	19.62	22.59	20.27	15.43	5.81
Miscellaneous	6.43	7.74	3.80	4.54	5.66
Other, common to all types	61.85	56.68	62.82	63.23	81.80
Total	100.00	100.00	100.00	100.00	100.00
Non-operating time percentage of total time on shift	22.45	27.06	26.52	26.69	29.25

Comparisons of Types of Shovels

These comparisons are made from data accumulated over a period including all seasons and of sufficient tonnage to be considered representative. While we may theorize on probable reasons for the differences, the actual use of the different types at this mine has shown the increased capacity of the shovel, equipped with caterpillars and 4½-cu. yd. dipper.

The electric shovel at the Utah Copper mine simplifies problems of fuel supply, which in turn is reflected in locomotive-haulage costs because of the elimination of interference with ore and waste trains caused by transportation of coal on the switchbacks and benches. Also, the electric shovel minimizes the problem of water supply and its power cost is small. As noted before the larger part of the year is winter, when it is difficult to keep the water lines open.

Figures prior to the advent of the electric shovels show that 269. 77 cu. yd. of material were excavated per ton of coal consumed and that 119.02 cu. yd. were excavated per 1000 gal. of water consumed. During a year with a production of that of the year just passed (viz. 1924, 12,126,600 tons of ore and 6,234,912 cu. yd. of stripping), the coal consumption was 104,615 tons and the water, 239,448,000 gal.

When comparing the outputs of the different shovels, it must be understood that while the figures were taken for as long a period of time as was possible, they pertain only to the particular operating conditions at this mine as they were during that period; it does not necessarily follow that they would apply at any other place unless all the controlling conditions were the same. In other words, it is impossible to segregate the shovels as an individual part of the machinery of the mine in order to judge their performance, but rather they should be considered merely as a link in the entire chain of mining operations. Also, it must be remembered that the capacity of a shovel does not depend solely on its own characteristics, as there are other factors that affect its performance. It is never possible to operate a shovel at full capacity for any extended length of time, so that any comparison made between one shovel and another must take into consideration all the conditions under which they operate, as well as their maximum full-time capacities. One factor that has an important bearing on the output of the shovel is the facility with which the loaded cars can be hauled away from the shovel site and empty cars substituted; the rapidity with which this can be done is beyond the control of the shovel operator.

Although at the Utah Copper mine performance has shown that the electric shovel has about the same digging capacity as the steam-shovel similarly equipped with tractors and dipper, the former has the advantage of lower cost in direct operation, in addition to the indirect saving just

stated. This amounts to a saving of 23.7 per cent. of the operating labor cost of the steam shovel and 79.6 per cent. of the operating material.

The many factors that comprise maintenance costs make a comparison difficult, and likely to be misleading, but it is thought that the electric shovel may have an advantage in this respect because the principal source of its electrical trouble is contactors and loose connections. These troubles seldom cause a delay of more than 30 min. and can often be repaired temporarily in much less time, and made permanent after the regular shift is completed. The electric shovel is always more clean and orderly than the steam shovel; this feature tends to raise the efficiency of the shovel crew. The absence of smoke and steam makes the electric machine safer, as the bank and track are always visible. The absence of smoke and steam on the benches, especially during cloudy weather, is an advantage to train crews because visibility is increased and greater train speeds are realized with safety.

Electric Equipment

At the Utah Copper mine both alternating- and direct-current equipments are used with satisfactory results. The alternating-current equipment consists of the following, located directly on the shovel:

One three-phase, 300 kv.-a, 5000/460-volt transformer, controlled through a suitable oil circuit breaker.

Two 100-hp., 440-volt, phase-wound, induction motors, for operation of the hoist motion.

One 60-hp., similar type motor for the operation of the swing; one similar motor for the operation of the crowd.

All of these motors are controlled with electro-pneumatic contactors actuated by proper sequence switches through drum controllers.

The direct-current equipment consists of the following:

One 225 kv.-a, three-phase, 60-cycle, 5000-volt, 1200-r.p.m. synchronous motor, controlled with a 7500-volt, solenoid-operated, automatic circuit breaker, and direct-connected to 125-kw., 250-volt, differentially compound-wound, direct-current hoist generator; 30-kw., 250-volt, differentially compound-wound, direct-current swing generator, and a 29-kw. 250-volt, differentially compound-wound, direct-current crowd generator.

Two 105-hp., 250-volt, direct-current, series-wound, mill-type motors for hoist service.

One 43-hp., 250-volt, direct-current, shunt-wound mill-type motor for swing motion.

One 40-hp., 250-volt, direct-current, shunt-wound, mill-type motor for crowd motion.

The control for these motors and generators consists of field resistors, with drum controllers for varying the field strength.

The four principal items affecting the selection of electrical equipment for use in the operation of shovels are: cost of equipment, power consumption, power factor, and maintenance costs. The cost of alternating-current equipment is approximately 18 per cent. less than direct-current equipment, and the maintenance cost of alternating-current equipment will be approximately 30 per cent. lower than direct current equipment, based on the ratio of rotating parts of both types of equipment. The power factor of the direct-current equipment is 54 per cent. higher and the power consumption approximately 15 per cent. lower than the alternating-current shovel.

Power is delivered at 44,000 volts, three-phase, 60 cycles, to two outdoor substations located at each end of the property (Figs. 15 and 16), each having a capacity of 2400 kv.-a. The power is transformed from

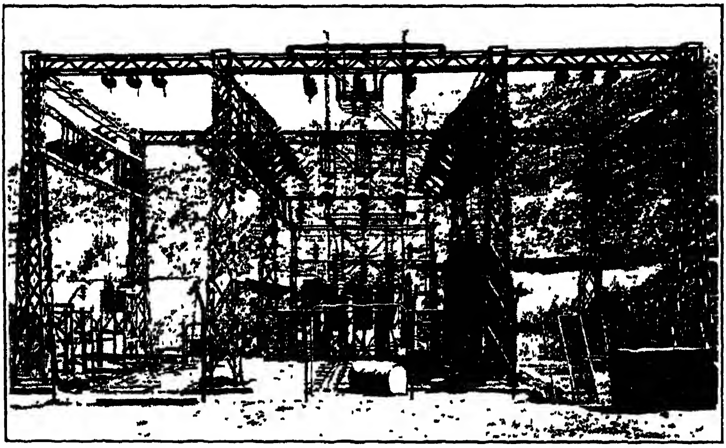

Fig. 15.—Outdoor substation.

44,000 to 5000 volts and delivered to two parallel lines extending around the mining area and having a length of about 5 miles. Taps are taken off to each of the benches through an automatic oil circuit breaker located in a steel switch house; Fig. 17. It is then transmitted along the benches over aerial lines supported on portable steel towers. Junction boxes are provided on the base of these towers, with necessary disconnecting switches to provide for delivery of power from either end of the benches, as may be required by the location of the shovel. A 500-ft. three-conductor, rubber-covered cable is used to transmit the power from these junction boxes to a suitable take-up reel on the shovel; Fig. 18. This trail cable permits the moving of the shovel 400 ft. across the bench; the cable is then disconnected and transferred to the junction box on the next tower. Normally, the power is delivered from in front of the shovel, which permits the disconnecting of the line in the rear of the shovel, which is in the blasting area, thus protecting service to the shovel.

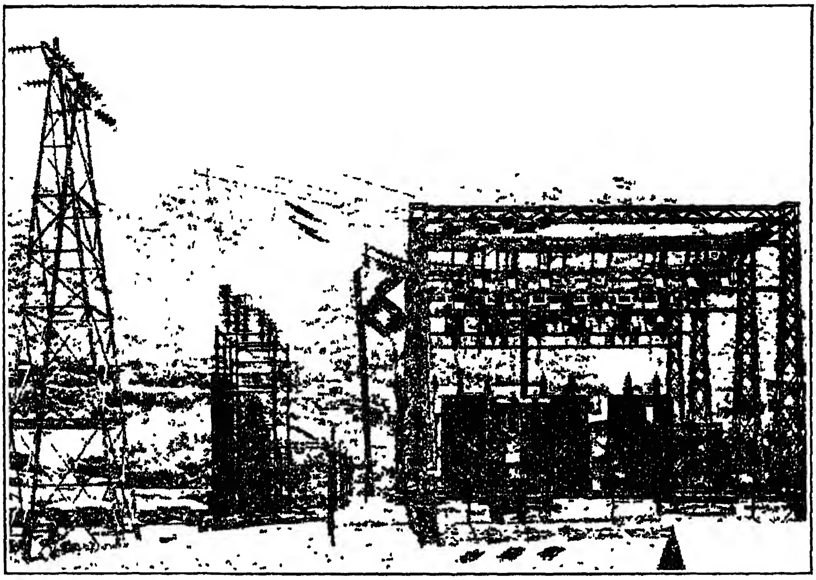

FIG. 16.—OUTDOOR SUBSTATION.

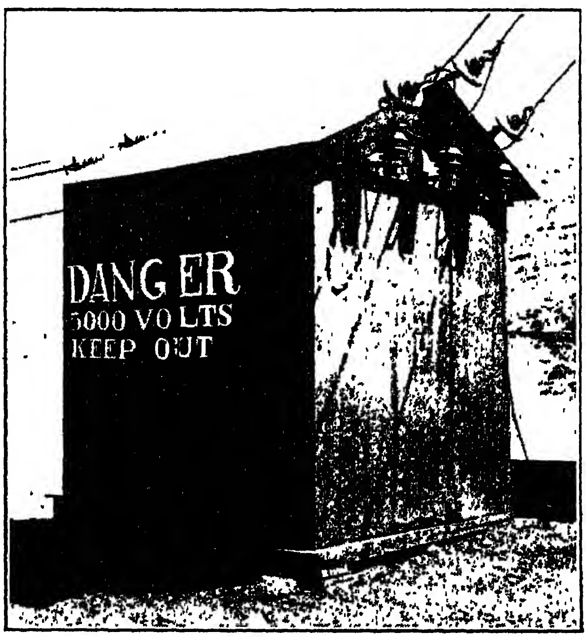

FIG. 17.—STEEL HOUSING OF OIL CIRCUIT BREAKERS.

The portable steel towers were designed to meet the requirements and are approximately 25 ft. high, with a top section 1.5 ft. square, spread to 6 ft. square at the base. The base is made of angle irons with turned up ends, so that the towers may be slid into place when it is necessary to move them.

FIG. 18.—TRAIL CABLE, SHOWING TAKE-UP REEL.

The assembled tower weighs approximately 900 lb. and is designed to handle three, No. 2, medium hard-drawn, stranded copper wires, in spans up to 300 ft. Each line is dead-ended through 11,000-volt electrose insulators on each tower; these insulators are used on account of the resistance of the composition to flying rocks due to blasting. This

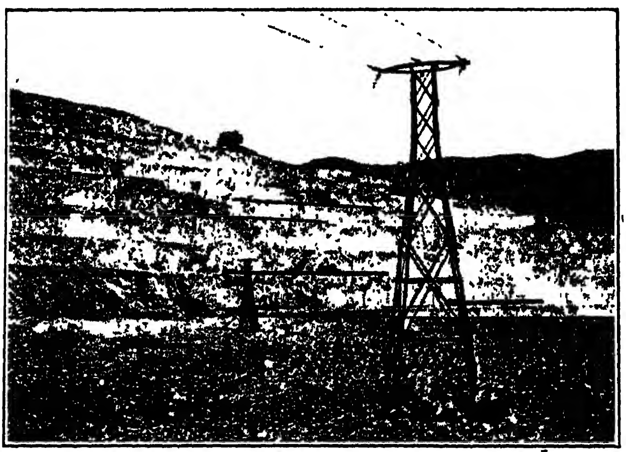

FIG. 19.—TRANSMISSION LINE OF PORTABLE BENCH TOWERS.

permits the removal of a damaged section of the line with a minimum delay. Towers are also designed to accommodate a double set of disconnecting switches, for the purpose of sectionalizing the lines and serving as a trail-cable connection to the shovel. A typical tower and the general appearance of the tower line are shown in Fig. 19.

A 5000-volt distributing system was adopted after a thorough investigation of the power to be delivered and the area to be covered, and proved to be the most economical distribution voltage for this particular installation. Some trouble was experienced at first from the effects of corona in the three-conductor trail cables; but this has been eliminated by changes in the manufacture of the cable. Permanent grounds are maintained and ground wires run parallel to all power circuits, to which all cable sheaths and shovel equipment are connected. A complete installation of lightning arresters is installed for the protection of the equipment.

The power required for the operation of electric shovels depends, to some extent, on the type of equipment used, whether alternating or

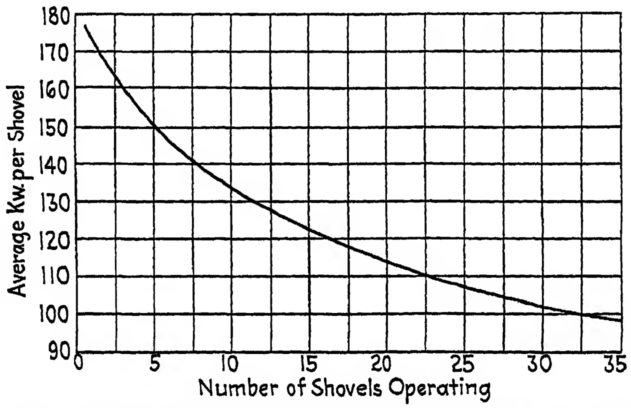

FIG. 20.—THEORETICAL DENSITY CURVE OF ELECTRIC-SHOVEL LOAD.

direct current, the average demand per shovel being 100 to 125 kw., while peaks will run between 250 and 450 kw. A number of variables, such as condition of bank, amount of moving and car supply, influence the power demands. When a group of shovels are operating, the diversity of the load between them must be considered when determining the ultimate substation capacity. This may be obtained, approximately, from a theoretical diversity curve, such as shown in Fig. 20.

As the use and operation of the electric shovel have proved satisfactory and economical, the company plans the complete electrification of all steam shovels.

ACKNOWLEDGMENTS

I hereby with pleasure acknowledge my indebtedness to, and take this opportunity to thank Messrs. J. D. Shilling, Louis Buckman, C. W. Corfield, R. E. Corfield, H. R. Sadd, and A. E. Robinson of the personnel of this organization, who have kindly assisted me in the preparation of this paper.

SAMPLING AND ESTIMATING

Sampling and Estimating Ore Deposits

SAMPLING, including assaying or analysis, and estimating of an ore deposit are necessary in order to determine something as to the form, size, characteristics, and value before any mining and treatment methods can be chosen, and an estimate of profits can be made. The degree of completeness to which this estimate is carried varies greatly. The low-grade disseminated copper deposits are thoroughly sampled before plans for development and production are started; if orebodies are less regular and continuous, the preliminary work is less thorough; if orebodies are in limestone the best that can be done is to prove that a certain area of this rock contains orebodies of a size and grade that can be found without excessive cost for development.

The methods of sampling and estimating vary with the types of deposits and often with individual orebodies. No rules can replace the judgment of the engineer, yet a knowledge of the methods employed in similar deposits will greatly lessen the task of planning the sampling and estimating of new orebodies. Further, figures that show how actual mining has checked other estimates make it possible for an engineer to accept his own estimate with more confidence or to make proper allowances or correction factors for errors that cannot be avoided in the type of orebody he is examining.

METHODS AND FACTORS

Geology

The importance of structural geology, surficial and at depth, is recognized by many engineers. At Butte, sampling and estimating prior to development is closely related to geology. Here and at Jerome, the geological departments of certain companies direct most of the sampling. At Bisbee, all workings in some mines are studied periodically by a geologist. Particularly for the disseminated or porphyry copper deposits and iron ores of Lake Superior is a good understanding of the geologic occurrence of importance. As a rule, the geologic features of ore deposits are noted on assay maps and are considered in estimating quantities.

Sampling from the Surface

According to whether the country rock or ore is hard or soft, diamond drills or churn drills are used, of course only for certain types of deposits.

At Butte, Cripple Creek, and on the Mother Lode, drilling has not been found to be satisfactory for exploration, but at Bisbee, Miami, Lake Superior (iron), Mineville, and Mascot, it is of value. In some cases especial care is devoted to the examination of cores and in others the sludge is carefully sampled, both as a rule by 5-ft. advance. What has to be contended with in drilling is caving of the hole, deflection of the bit, and concentration of mineral at the bottom of the hole.

For underground sampling, it has been found at Jerome and Kennecott that the diamond-drill cores are useful in new ground, and in sulfide areas at Jerome the core assays check closely with drift development. At the Homestake, one or two drills are constantly employed. For sampling lead-silver replacements in limestone, machine-drills are advocated.

Ore Sampling Underground

Where the pick or moil is used the cuttings are caught in a sack, box, or pan; grooves are mostly at 5-ft. intervals. At Butte, on the Mother Lode, and Jerome, only pick samples are taken, although at Jerome a few grooves or channels are cut. Regular moil sampling is done at Beatson, Copper Queen, Miami, Cripple Creek, Elkoro, Zaruma, Lucky Tiger, Mogollon, and Lake Superior (iron); and at Calumet and Arizona, Homestake, and Silver King Coalition for the purpose of estimating reserves or in new ground.

In spite of all that has been said and written against the practice, grab sampling gives satisfactory results after blasting at Iron Cap; from mine cars at Calumet and Arizona, Copper Queen, Homestake, Elkoro, Mogollon, Mother Lode, Hecla, and Silver King Coalition; from chutes at Calumet and Arizona, Copper Queen, and Cornucopia; and from "muck" at Juneau, Cripple Creek, and Mascot. No sampling is done at Copper Range because of the metallic copper in the ore. No regular sampling is done at Kennecott, Juneau, Hecla, Bunker Hill and Sullivan, Mineville, or Mascot, because as in the oxide and carbonate ores of Copper Queen, experience with the material is a good guide as to what is ore—in other words, visual examination is as good as sampling. At Bunker Hill and Sullivan, as long as galena is present, the ore is mined. Experience at Miami indicates that in moil work the cutting can be done too carefully, without greater accuracy of results.

Estimating

Regularity and uniformity of orebodies and mineralization tend to accuracy in sampling and estimating, other things being equal; irregular deposits and mineralization are difficult to sample and estimate. In all cases, there are correction factors to consider; these are based on experience, mill results, correct weight of ore in place, dilution of ore with

waste, or loss of ore in mining. The correct interpretation of data shows the skill of the engineer.

At Butte, the general accuracy of estimates by pick sampling varies considerably, although large-scale checks have agreed within 0.2 per cent. copper and silver.

At Beatson, where double cuts are made, the estimates are 20 per cent. high.

At Kennecott, the mine samples are higher than mill recovery, because of the character of the ore.

At Copper Queen, grab and groove samples result in a difference of 0.2 per cent. in copper between mine and smelter.

At Miami, where channel sampling is the practice, several million tons of ore milled show that the tonnage estimated was exceeded but the copper content was lower. Grab samples are also taken and average within 8 per cent. above the mill sampler.

At Juneau, grab sampling gives a close estimate.

At Homestake, grab sampling and channels are reasonably close to the actual recovery.

At Cripple Creek, the Portland mine with channel sampling reports a tonnage 20 per cent. in excess of estimates, because the stopes break wider.

At Elkoro, channels give a value 17 per cent. higher than the true value.

At Telluride, the tonnage exceeds the estimate, sometimes by 40 per cent., even though moil samples are taken.

At Zaruma, the tonnage is exceeded and the value is less than the estimates.

At Lucky Tiger, the silver is underestimated by 8 per cent.

At Mogollon, channel sampling and mill results agree within 3 or 4 per cent.

Along the Mother Lode the estimates are 10 to 20 per cent. high in value and low in quantity.

At Lake Superior (iron), there have been large discrepancies between careful estimates and ore extracted, yet some deposits have exceeded the estimates.

At Silver King Coalition the grab samples are from 5 to 10 per cent. higher than results given by the Sampling mill.

In effect then, it may be stated in general terms that the methods of sampling and estimating vary with local conditions. In spite of the greatest care, there is often considerable difference between the estimated and the actual content of an ore body. No one method can or should be set down for all types of deposits. All that can be done is to rely on experience with a particular orebody, and sample and estimate in the light of that experience.

Summary of Practice in Sampling and Estimating

While a certain procedure in sampling is more or less common to all types of deposits, these examples show specialization in each case depending on the mineral and the type of orebody, and therefore the methods have been classified under six heads; namely, copper, gold, silver-gold, iron, lead-silver, and zinc.

Copper Ores

Fault Fissures.—Although no elaborate system of sampling is claimed for the Iron Cap mine at Copper Hill, Ariz., Lees (see page 373) states that blocks of ground that have been mined have checked within 0.5 per cent. on both tonnage and value reported. The ore is a brecciated shale or quartzite with iron oxide and copper sulfides. The ore is from 3 to 40 ft. wide. Grab samples only are taken from each round blasted. A record is kept of all samples taken in the block of ore lying between any two raises, and the numerical average for the month is assumed to represent the value of the ore mined that month.

Reserves are estimated once a year. The area of each block of ground remaining between any two raises is measured on the profile tracings with a planimeter. This area times the average width of the block gives cubic contents, and this times 11 (cubic feet of ore in place in 1 ton) is the tonnage for that block. The numerical average of the year's samples of ores broken, together with the assay values of the drift over this section, is reported as the assay value of the block remaining.

Fissure Veins of Complex Systems.—The ore deposits at Butte carry copper, zinc, silver, and manganese, but only the copper-silver and zinc-silver ores will be considered here. According to the Committee on Mining Methods (see page 239) for this district, there is some difference between the estimated grade of ore blocks and the grade of the ore actually mined, but these differences depend largely on the width and character of the veins; these characteristics are known from the geologic notes. Where the ore is uniform and the walls are definite, the methods of sampling and estimating described are very accurate, and in a check on the grade of ore shipped from several drifts against the estimated average for the same drifts an error of only 0.5 per cent. was found in both copper and silver. In general, the accuracy of tonnage estimates varies considerably, although in many cases the figures have been close to the final result of mining.

At Butte, sampling and estimating preliminary to development is closely related to geology. Churn or diamond drilling has not been found to be satisfactory for exploration. Test-pitting and trenching and tunneling are seldom done to determine mineral contents of orebodies, rather to seek further geologic evidence. The geological department

of one company operating a large number of mines directs all sampling. The cut, or channel, across the face of drifts is impractical here because of the time consumed, so samples are cut by means of hand picks and caught in 6 by 14-in. sacks. This method is employed by the samplers of four other companies. There is no actual check sampling on the surface against specific underground sampling. As a rule, two samplers are employed at each mine. A sixth company samples the broken ore from certain workings as it comes to the surface. Records of where the samples were taken and the assay results are copied into loose-leaf ledgers, on which curves are plotted. When these cards are placed under corresponding geologic maps, the latter become assay maps and are used in estimating ore reserves.

With the exception of one company, the estimation of tonnage is similar. In general, a longitudinal section of each vein is made on a scale of 1 in. to 100 ft. and projected on a vertical plane parallel to the strike of the vein. A certain minimum width and assay value are assigned as the lower limit for ore. The various drifts and raises are given different colors, which show what should be included in the ore reserve for each section. Blocks are laid out and their areas determined by a planimeter. The best average relation of volume to tonnage is 10 cu. ft. to 1 ton; this varies not less than 9 and not more than 11. Of course, width, pitch, rock inclusions, and other irregularities are considered in the estimates.

Lenticular Deposits.—In the Beatson mine on Prince William Sound, Alaska, the ore consists of chalcopyrite associated with pyrite and quartz in a shear zone of graywacke and slate. The hanging wall is well defined but the foot wall is not. Some diamond drilling has been done but drifts and crosscuts are driven to explore most of the ground. When sampling these workings, as well as raises, a double groove 1½ in. deep and 6 in. wide is cut in each working place at 5-ft. intervals. According to Birch (see page 149) the results, even with such large samples and double sampling, are 20 per cent. too high. When estimating tonnage 12 cu. ft. is taken as equal to 1 ton of ore.

Schist Replacements.—The main orebodies developed in the United Verde, United Verde Extension, and Copper Chief-Iron King mines at Jerome, Ariz., are nearly massive sulfides in which pyrite greatly predominates although chalcopyrite is the most important mineral in the first and last named and chalcocite in the United Verde Extension mine. The main sulfide masses are of irregular pipelike form.

At the United Verde, according to Mills (see page 595), although diamond-drill cores from schist areas cannot be relied on for accurate data on the mineralization, they are useful in determining favorable geologic formations and the position and size of mineralized areas. At

the same time, the assays of cores from the massive sulfide areas check closely with subsequent drift development.

Actual sampling of stopes and development in mineralized areas is performed by two men in charge of the geology department. As a rule the method is to take a 5-lb. sample with a prospector's pick. Sometimes a groove or channel sample is taken. In stopes, each sample represents a block 5.5 by 5.5 by 7.17 ft., or what is termed a "set." This set is the unit of extraction and all engineering records on stopes give the tonnage in sets rather than in tons. Three samples are taken in drifts and crosscuts at 5-ft. intervals at left, face, and right, except where the formation is known to be valueless.

Replacements of Limestone.—Under this head are three important deposits, widely separated, at Kennecott, Alaska, and Bisbee, Ariz. In the three mines at Kennecott the principal mineral is chalcocite. There are no defined walls to the orebodies.

Birch (see page 499) states that at Kennecott it is impossible to foretell or estimate accurately the tonnage or grade of ore that a block of ground will produce without an unreasonable amount of development work. Although diamond drilling has been employed to good advantage in exploring unknown areas, drifts and crosscuts are the most reliable methods. After becoming acquainted with the ore, it is possible to estimate its grade closely by the amount of glance or carbonates contained, so no regular sampling is done underground. Mine samples generally are considerably higher than the actual recovery in the mill, due probably to friability of the glance and the soft, chalky nature of some of the carbonates.

Experience at the Calumet and Arizona mines, at Bisbee, shows that limestone replacement and contact metamorphic orebodies are so irregular that sampling and estimating are far less exact than in more regular types of deposits. According to Dickson (see page 621), all workings are studied periodically by a geologist. Drifts, crosscuts, and raises are sampled by channels or by taking a grab off each mine car as it is filled. A chute sample is the best for stope ore. Geologic data, characteristics of veins, classes of ore, and assays are transferred to a set of maps and reserves calculated therefrom.

In the Copper Queen mines, the replacements of limestone are irregular and mainly consist of oxide and carbonate ore, according to Prouty and Green (see page 628). All available geologic information must be used in sampling and estimating. Underground, four classes of samples are taken: (1) grab samples taken by the shift boss or foreman and assayed by the potassium-cyanide method; (2) chute samples taken from cars as loaded by car men under shift boss orders, and assayed by the cyanide method; (3) belt or shipment samples from the loading belt at the main shaft and from mine cars, when dumped, for the ore that is not hoisted;

and (4) channel or groove samples taken by the sampling department for estimating the grade of ore developed before stoping commences. The last two classes of samples are assayed by the potassium-iodide method.

Little channel sampling is done in oxide and carbonate ores because the change from ore to waste is easily recognized by an experienced man, but sampling is essential in sulfide ore. The average difference between chute and belt samples is about 0.4 per cent., and between the shipment samples taken at the mine and smelter about 0.2 per cent. on a copper content of about 7 per cent.

Replacements of Porphyry.—In the replacements of porphyry at Sacramento Hill, Bisbee, the boundary as a rule cannot be determined without sampling, according to Prouty and Green (see page 635). There is a gradual decrease in mineral content until barren material is reached; this gradation may occur within a short distance or it may extend for several yards. In sampling this deposit, churn-drill holes were put down at the corners of 100-ft. squares. Samples of sludge representing 5-ft. intervals were passed through a Jones splitter. Three cuts removed one-eighth as a sample which was dried and assayed. As the drill holes pierced underground workings, raises were put up along the holes and the ore was sampled, and a comparison of eighteen samples by each method gave 3.95 per cent. copper for the drill and 3.77 per cent. for the channels. Final estimates were made by the triangular-prism method.

Replacements of Contact Breccia.—This type of deposit occurs in the Warren district, Arizona, and is extremely irregular in position, size, and copper content. It offers one of the most difficult problems in prospecting and without sampling it is impossible to determine whether the sulfide masses within are ore or waste, according to Prouty and Green (see p. 629).

Disseminated Copper Metal.—In discussing operations at the Copper Range mines, near Houghton, Mich., Schacht (see page 348) states that the copper deposits of the Lake Superior district are most difficult to sample and evaluate, and it is impossible to sample the rock in the manner generally applied elsewhere because of the non-uniformity of the copper. A mill test is really the only satisfactory method.

Disseminated or Porphyry Copper Deposits.—According to Joralemon (see page 607), the sampling of a disseminated copper orebody is simple and should be very accurate. The estimation of tonnage and grade of ore in the deposit, once the length and grade of ore in every opening have been calculated, is a simple mathematical problem. The combination of individual assays in a drill hole or other opening to give the length and average grade of ore that will be mined is the difficult part of the problem. To solve it, the engineer must have an intimate knowledge of the geologic occurrence of the ore, of the mining method to be employed, and of the probable costs of operation. The final estimate should check the tonnage mined within less than 0.1 per cent. copper.

At the Miami mine, Arizona, according to Hensley (see page 79), where sampling is done by churn and diamond drilling and by channels, all at 5-ft. intervals, the following results were obtained:

Case No.	Expectancy		Actual Extraction		Extraction, Per Cent.		
	Ore, Tons	Copper, Per Cent.	Ore, Tons	Copper, Per Cent.	Tonnage	Grade	Copper
1	3,772,254	1.959	3,954,320	1.763	104.8	90.0	94.3
2	2,464,303	2.190	2,483,581	1.811	100.78	82.7	83.34

No great care is exercised in cutting the channel samples as experimental sampling indicates that no greater accuracy is obtained and considerably more time is consumed.

Grab samples are also taken from each haulage car as it is loaded from the transfer raise. They are assayed daily and a correction factor is applied at the end of each 10-day period. The grab samples agree within an average of 8 per cent. above the final result obtained from the automatic sampler in the mill.

Gold Ores

The types of gold ore deposits described may be divided, for the purpose of discussing sampling, into lodes and veins.

Lodes.—At Alaska Juneau, according to Bradley (see page 103), gold occurs erratically and is mostly in quartz stringers and gash veins in slate and metagabbro; it is unusually coarse for lode gold. In general, it is difficult to determine in advance by any system of hand sampling the average gold content of this ore. At Alaska Juneau the selection of the sampling method to be used is one problem for the management but another for an examining engineer. Systematic channel sampling is considered unnecessary. A knowledge of ground not already known through actual mining is gained by grab samples taken from the muck during the progress of development work. The assays of these samples are interpreted in the light of experience and knowledge of the ground. The chief purpose of sampling, at Juneau, is to determine the grade of what the miner recognizes to be ore and to make a permanent record of the information. The selection of a man suitable for grab sampling requires care. At the Treadwell group of mines, opposite Juneau, grab sampling proved to be very satisfactory. Over periods of 14, 15, 16, and 17 years the grab samples at four mines averaged \$2.20, \$2.72, \$2.88 and \$2.37 per ton, and the mill recovery plus tailings was \$2.33, \$2.24, \$2.83, and \$2.47, respectively. The real problem in sampling and estimating

is not only obtaining data but interpreting the data; correct interpretation is the real test of the ability of the engineer.

At Homestake the ore is wide, hard, and tough. One or two diamond drills are constantly at work exploring new ground. Channel samples are taken where development work is in ore, and grab samples from ore cars drawn from caved areas, when there is doubt as to whether it is ore or waste. Working and assay plans of each level, stope, and pillar are kept up to date. Ten cubic feet of ore in place equals one ton. So far, the estimates have been reasonably accurate, although dilution by waste from caved areas of old workings has increased the tonnage and reduced the grade, according to Ross and Wayland (see page 426).

Veins.—Briefly, the orebodies at Cripple Creek consist of enriched areas along gash veins and dikes that traverse a volcanic plug of breccia. As the oreshoots are very spotted, diamond drilling is not considered the best method of exploring the ground, according to Jones (see page 512). Stopes are from 3 to 10 ft. wide. When estimating the value of a block of ore, samples are taken by shift bosses across the streaks, with an occasional sample of the adjacent rock, and grab samples from the muck pile. Tonnages are calculated on a basis of 13 cu. ft. of ore in place and 22 cu. ft. for broken ore. The quantity mined runs about 20 per cent. higher than the estimates in the Portland mine because the stopes break wider than expected.

In the Elkoro mine at Jarbidge, Nev., the gold and silver occur in well-defined quartz veins that vary greatly in size and value. The ore, according to Park (see page 518), is generally soft and estimates are based on 18 cu. ft. per ton in place. Most of the work is in oxide ore. Diamond drilling is unreliable. Drifts driven on the vein are sampled with the pick at 3 to 5-ft. intervals; the samples weigh from 10 to 12 lb. The face is sampled in three or more sections after each round; in crosscuts, a cut is taken continuously along the side. A grab sample is taken from each car of muck by trammers or shovelers; the samples thus obtained serve as a rough check on those cut from the face. A final car sample is taken from the train delivering ore to the outside orebins, which serves as a check on the mill heads. The estimated value runs 17 per cent. higher than the true value of the ore mined, so this factor is employed in reporting on new ground.

The veins of the Telluride district of Colorado are long and from 14 to 48 in. wide, according to Bell (see page 550). The ore is fairly hard. In addition to free gold and silver it carries many minerals. Cut samples are taken systematically at 10-ft. intervals and are averaged geometrically. Chute samples are taken if mill heads fall below a predetermined point. Assays are recorded on assay plans. Estimating is simple, but dilution is a factor. From 11¾ to 13 cu. ft. of ore equals 1 ton. The tonnage mined is generally more than that estimated, sometimes 40 per

cent. above. All assays over $20 had to be eliminated at one mine, while all high assays had to be included at another.

The gold-quartz veins worked by the South American Development Co., Ecuador, are in a belt of greenstone, according to Emmel (see page 453). Close sampling is practiced wherever a vein is exposed. Channel samples are moiled from the face and back 1 meter apart; about 15 lb. is cut per meter. If the character of the vein changes, the samples may be split. Stope faces are frequently sampled. Because of the indefinite walls at many places, the tonnage recovered is generally more and the value less than estimated from moil sampling.

California Mother Lode, according to Arnot (see page 290), may be considered a magnified stringer zone, with the stringers represented by the various quartz veins. About 70 per cent. of the gold is free and most of the remainder is in pyrite, with some in arsenopyrite and galena. Diamond drilling has not been a success as a means of exploration. Little sampling prior to shaft sinking has been attempted. Visual examination and panning have been the systems in vogue. As the vein walls are generally well defined and the gold is uniformly distributed, underground sampling has been carried on in a desultory manner. A few companies sample closely and have assay plans. Cut samples are easily taken with a hand pick; grab samples are taken from cars. As a rule, foremen and trammers do this, but at two mines samplers are employed and keep geologic data up to date. The grab samples taken underground are checked against grab samples taken at the mill. Estimating ore along the Mother Lode is generally a relatively simple matter. The ore averages 12.5 cu. ft. per ton in place. As a rule, estimates are 10 to 20 per cent. high in value and low in quantity, depending on vein width, because of dilution from the soft walls.

Silver-gold Ores

In the Cornucopia district of northeastern Oregon, the ores are quartz in irregular lenses. Inclusions of wall rock are common and are a factor in reducing the grade. In mining, the hanging wall slabs off and dilutes the ore by 8 per cent. Hand sampling of faces is not attempted but samples are taken from chutes and from broken rock in the drifts after blasting, according to Betts (see page 155).

Sampling is of importance at the Lucky Tiger mine in Sonora, because the veins only average 19 in. in width, but the stoping width is 39 in. Ore in place, which assays 73 oz. of silver per ton, only carries 40 oz. at the mill because of unavoidable dilution in mining; therefore, when sampling development work, dilution is accepted as unavoidable. Samples are cut over the full stoping width of 36 to 42 in.; if the vein is wider, the full width is sampled. Each channel across the back is divided into as many samples as there are varieties of ore and waste, but as a rule only

two samples are taken—one of ore and another of waste. Drifts, raises, and winzes are sampled at 5-ft. intervals. Backs of stopes are similarly sampled whenever chute samples show that the ore is lower than milling grade. In shrinkage stopes, weekly samples are taken across the fill at 10-ft. intervals. Grab samples of two double handfuls are taken from each car of ore as it is loaded. Chute samples are taken daily; some are assayed daily, others every two or three days. As to the accuracy of the sampling, Mishler and Budrow report (see page 476) that during a period of 14 years the estimated ore reserves averaged 34.0 oz. of silver per ton and the ore mined averaged 37.1 oz. This is an underestimate of 8.4 per cent., which is not unsatisfactory for grab samples. An assay map is maintained and the tonnage factor is 11.5 cu. ft. of ore per ton.

Mogollon.—The quartz-calcite veins of the Mogollon district of New Mexico lie in hard rock and vary from 2½ to 4 ft., 8 to 15 ft., and 20 to 24 ft. in width, according to Kidder (see page 529). Samples are taken by trammers on each shift from cars from every development face. Later, to prepare assay plans and estimate reserves, moil samples are cut at intervals of 5 or 10 ft. according to the character of the ore. Winzes and raises are cut at 5-ft. intervals and backs of stopes every 10 ft. Chute samples are also taken. Sampling and mill results seldom vary more than 10 per cent.; they are generally within 3 or 4 per cent. In two lots of ore of 51,862 tons and 43,993 tons, the mine and mill sampling agreed as to value within 1.20 per cent. and 9.17 per cent., respectively.

Iron Ores

Bedded Deposits.—At a meeting of the Mining Methods Committee at New York on Feb. 16, 1925, W. R. Crane of the U. S. Bureau of Mines briefly told of the sampling he had done in the iron mines of the Birmingham district. The Big Seam is from 15 to 22 ft. thick but is divided into two benches of about 10 ft. and 8 ft. by slate up to 30 in. thick (see page 158). The dip is about 16°. The ore averages 36 per cent. of iron and is remarkably uniform as it does not vary more than 2 per cent. in 15 miles along the bed. The lime content is about 17 per cent. and the silica 11 per cent. Some discussion arose among the operators as to the variation of the ore in depth, so Mr. Crane sampled one portion over a mile in length. Sampling was done with hammer and moil and a groove was cut from the top to the bottom of the bed exposed. The cuttings were caught in canvas and averaged 10 lb. per sample; no crushing or quartering was done until the samples reached the laboratory. Results showed that there is a gradual though slight decrease in iron content down the dip. This sampling is bearing out Mr. Crane's original sampling.

Interbedded Deposits.—Wolff, Derby, and Cole (see page 641) describe the sampling and estimating of Lake Superior iron ores as a whole. Exploration is done principally by drilling—churn drills and sludge for

the soft ores and diamond drills and cores for the hard ores. In drifts and crosscuts, the ore is sampled in 20 or 25-ft. sections and each section is sampled at four stations. Grooves 3 in. wide and 2 in. deep are cut across the entire face of the ore at right angles to the formation, or grooves are cut between each set of timbers, 5-ft. intervals, on opposite sides of the drift. The cuttings are caught in a pan or box. Raises, winzes, etc. are sampled in 5-ft. sections where the ore is uniform; but if rock or lean ore occurs with the ore in the same section, samples are taken of each material. When estimating, structural geology must be considered and the cross-section method is employed.

In describing the mining methods of the Michigan (Marquette Range) district Elliott, Jopling, Chenneour, and Derby (see page 657), treat on sampling during exploration by diamond-drilling; while in describing sampling Marquette ores, Bowers (see page 657) treats on underground and top landing samples. When drilling from the surface, the only sampling done is that of the core and sludge after each 5-ft. run. Core is obtained from hard ground and sludge from soft ground. A factor of 8 or 9 cu. ft. per long ton is used for hard ores, 12 cu. ft. for soft hematites, and 13 or 14 cu. ft. for limonites. As the deposits are so deep and irregular, there have been large discrepancies between careful estimates and the ore eventually mined; although some deposits have proved to be much greater than estimated.

Underground, in order to hoist the ore according to grades, stope samples are taken. Grooves are cut by pick across the breast at right angles to the dip of the ore. Chute samples are taken by means of a scoopful (½ lb.) from mine cars. On the surface, fifteen scoopfuls, or about 8 lb., of ore are taken from the tops of railroad cars at regular intervals diagonally from end to end. From each dipper of the steam shovel at the stockpile one or more scoopfuls are taken. Skip samples are taken with a scoop or dipper at the pocket, from each skip loaded into the railroad cars. Samples from all of the mines are accumulated in units of from one to twenty cars, depending on from how small a unit of loading the analysis is required. Scoop samples are not taken haphazardly but the points of sampling are determined by means of a knotted rope (clothes line) thrown or laid across the tops of mine cars or railroad cars. Where the knots lie the samples are taken, which system gives little chance for selection by the sampler.

Lenticular Deposits.—The diamond drill is employed at Mineville, N. Y., to explore outcrops, to determine the extent of ore being worked, and to "feel" out an orebody in advance of a proposed slope. Great importance is attached to the core for visual and chemical examination. Sludge samples are generally 5 per cent. higher in iron than the cores. Mine sampling (that is, cut samples) is seldom done; inspection of working faces gives a close approximation of the grade. The crude ore sent

to the mill is sampled daily by passing a pan every half hour through the stream of ore after it has been crushed to 4 in., according to Henry (see page 228).

Lead-silver Ores

Lode and Vein Deposits

According to McCarthy and Foreman (see page 319), the orebodies of the Hecla mine at Burke, Ida., are from 3 to 40 ft. wide and are mostly vertical. Milling ore is sampled at the head of the mill by an automatic sanpler of the Vezin type. No attempt is made to sample the crude ore at the mine; the smelter sample is used in all cases. Certain sections of the mine, where the assay is low or variable, are sampled every carload and the sample is assayed every week. Grab samples are occasionally taken by the mine foreman for his own information. As the result of chute sampling for 3 months throughout the mine, it was found that when compared with the metallic units hoisted (smelter assays, mill heads, and waste) these samples were higher by 12.401 per cent. silver and 10.761 per cent. lead.

The ore deposits of the Bunker Hill & Sullivan mines at Kellogg, Ida., according to Childs and Easton (see page 306), consist of large irregular masses of galena with siderite and quartz or of well-defined veins. No mine sampling is done. All material that shows galena is mined and enough broken waste is sorted out to bring the grade up to mill feed quality. The silver ratio is constant, so sampling for this metal is unnecessary. Where the ore is actually sampled is at the mill and the heads are of utmost importance in indicating the grade of current production.

In the Silver King Coalition mine at Park City, Utah, there are both lode deposits and replacements in limestone. According to Lewis (see page 485), most of the samples are grabs from cars underground. The assays from these samples are from 5 to 10 per cent. higher than those from the sampling mill. Groove samples cut with a pick are taken in new stopes. It is seldom necessary to sample the first-class ore, so most of the samples are of second class or milling ore.

Replacements in Limestone.—According to Prescott (see page 666), a thorough knowledge of the peculiarities of replacement deposits is necessary in sampling. Oxidized ores are often overestimated because of error in determining specific gravity and actual cross-section of the orebody; and sulfide ores are often underestimated because it is difficult to scatter the samples so that they will accurately represent the area under consideration. When sampling replacement deposits, the personal factor is considerable, and the result of sampling and estimating is more often an opinion rendered than a logical presentation of facts with supporting data. Determination of the form of orebody is essential, a record of

previous production is important, and there should be a correction for included limestone, which varies from a negligible amount to 30 per cent. of the material within the stope walls.

When cutting samples (larger than customary in other deposits) in oxide ore, care is taken to keep the channels even and at right angles to the banding. Sampling should consist of scattered and irregularly spaced cuts aimed at the determination of the ratio of the precious metals to the base metals, and the grade of ore that may be expected from the deposit rather than the grade of the material exposed.

The sampling and estimating of the sulfide bodies in replacement deposits present an entirely different set of conditions from those encountered in the oxides. The ores are often massive, sometimes fairly uniform in shape and in distribution of minerals, but more often they exhibit great irregularity. If the orebody has been prepared for stoping, channel samples intelligently laid out rather than mechanically spaced will give good results, but often better results are obtained by drilling the ground with machine drills and using the cuttings as the sample. Short holes are sufficient for this purpose. Careful mining often yields a better result from the sulfides than the sampling warranted, especially if a certain amount of selection and sorting is possible.

Zinc

The orebody at the Mascot mines, Tenn., (see page 54) is very irregular. Most of the ore mined consists of dolomitic limestone seamed with veinlets of dolomite and sphalerite, according to Coy and Noble. When churn drilling in ore, all bailings from each 3 ft. of advance are sampled and assayed; when diamond drilling in ore, all cores from each 5-ft. advance are ground and sampled for assaying. Grab samples are taken underground from the muck pile after each round is shot in ore being developed. Practically no other underground samples are taken. Because of the irregularity and nature of the mineralization, cut samples are not much more reliable than estimates, and it is comparatively easy to estimate grade closely on a freshly broken face. A factor of 12 cu. ft. per ton is used, but no statement can be made as to the accuracy of methods of estimating tonnages.

Discussion of Summary on Sampling and Estimating

To check up estimates of tonnage made in the opening of a mine has been found most difficult. Most engineers to whom this matter has been referred have found that larger tonnages have been mined, or are to be mined, than were originally estimated. This result is obtained mainly by improvements in the metallurgical processes and the possibility of treating a lower grade of ore.

Table 1 gives the earliest fairly complete estimates of the tonnage and grade of ore mined since these early estimates, and the estimated ore remaining in several big "disseminated" copper deposits. The figures were taken from annual reports. Discoveries of additional ore made while mining was going on make the figures inconclusive. But it seems clear that in the case of both open-pit and underground mines, if the geology and mining method are worked out in advance an estimate will show almost exactly what tonnage and grade of material will be mined. If the estimate is made according to arbitrary rules, without taking into account the geology of the deposit or the mining method, it will show much too high a grade and too low a tonnage.

TABLE 1.—*Results of Estimates in Low Grade Copper Mines*

Mine	Mining Method	Early Estimate		Ore Mined since Early Estimate			Estimated Ore Remaining	
		Tons	Copper, Per Cent.	Tons	Copper, Per Cent.	Percentage Difference in Grade from Estimate	Tons	Copper, Per Cent.
1	Caving	75,096,000	2.17	27,796,400	1.66	−23 5	78,762,296	2.07
2	Caving	20,800,000	2.48	20,536,819	2.05	−17 3	5,106,643	2.08
3	Caving	97,143,000	1.63	34,202,118	1.31	−19.6	62,695,910[a]	1.73
4	Caving	239,192,000	2.21	17,069,556	2.32	+ 5.0	258,940,000	2.24
5	Open pit	301,500,000	1.532	105,461,511	1.335	−12.9	347,378,049	1 35
6	Open pit	339,000,000	1.91	29,324,941	1.66	−13.1	327,629,889	1.91
7	Open pit	90,500,000	1.80	26,056,508	1.70	− 5 5	101,163,114	1.50
8	Open pit	40,360,823	1.70	38,531,947	1.60	− 5.9	60,680,660	1 62
9	Open pit	14,696,000	1.50	9,901,733	1.495	− 0.3	4,794,267[a]	1.51

[a] Estimate of ore remaining not published, so obtained by subtracting ore mined from last published estimate.

Methods of Sampling and Estimating Copper Deposits

The three papers following describe and discuss methods and results of sampling copper ore in disseminated deposits, in limestone replacement and contact metamorphic orebodies, and in replacements of limestone, porphyry, and contact breccia.

Sampling and Estimating Disseminated Copper Deposits

By Ira B. Joralemon,* A. M., Warren, Ariz.

(New York Meeting, February, 1922)

The sampling of disseminated copper deposits has been described often but the method of combining assays to give the true shape and value of the orebody as it will be mined has received less attention. As the ore mined during five or ten years at several of the great disseminated properties has averaged materially lower in grade than the published estimates, the question of the sampling and estimating of these orebodies seems timely. In this paper, attention is given particularly to factors that may cause errors in the final estimate. Sampling of diamond-drill holes, churn-drill holes, and underground workings is briefly described in the first part of the paper; then the method of combining assays to give the average grade and tonnage that will be mined is taken up in some detail. The necessity of study of the geology of the deposit and of probable costs is emphasized. Finally, the methods of calculating tonnages are outlined.

Diamond-drill Sampling

There are three main requisites for accurate diamond-drill sampling. First, the core should be removed and the hole cleaned out at the beginning of every sampling interval, usually 5 ft. (1.5 m.). After the rods are let down, water should be run through them until it becomes clear. The water must be clear before drilling starts. In this way the material knocked down from the sides of the hole by the descending rods is washed away, instead of being left to salt the sludge sample.

The second requisite is to save all the sludge and all the core from every sampling interval. Where the material is uniformly heavy, the cuttings will settle sufficiently in a box, through which the sludge flows slowly.[1] Where part of the material is too light to settle readily, the sludge should be run into barrels, and allowed to settle until the water becomes clear. Then the clear water is siphoned off, and the sludge evaporated to dryness over a slow fire, sacked, and assayed.[2] To obtain

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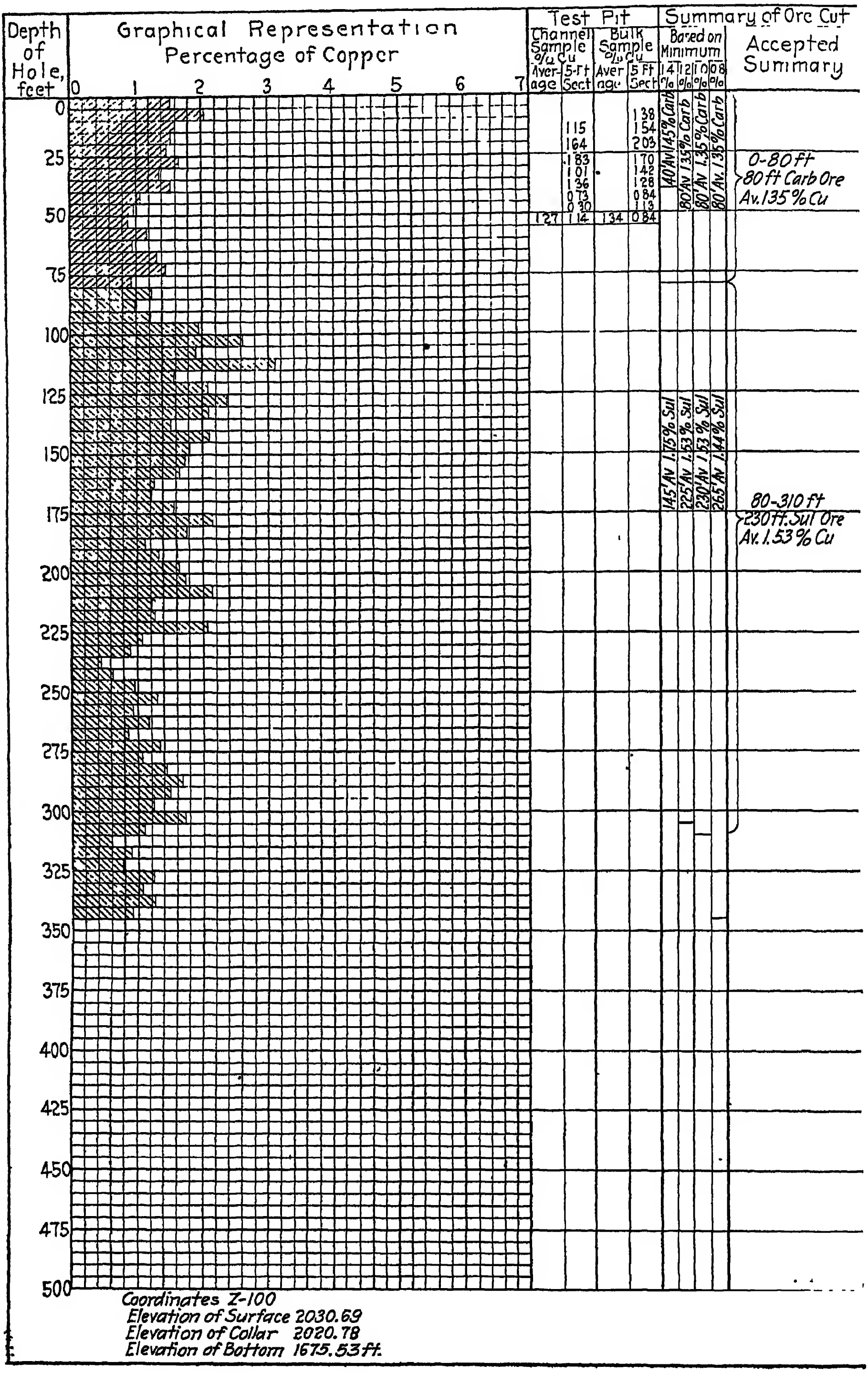
[1] E. E. White: Surveying and Sampling of Diamond-drill Holes. *Trans.* (1912) **44**, 69.

[2] Ira B. Joralemon: The Ajo Copper-mining District. *Trans.* (1914) **49**, 605.


Depth of Hole, feet
Graphical Representation
Percentage of Copper
0 1 2 3 4 5 6 7
Test Pit
Channel Sample % Cu
Aver-age
5-Ft Sect
Bulk Sample % Cu
Aver age
5 Ft Sect
Summary of Ore Cut
Based on Minimum
1.4 % 1.2 % 1.0 % 0.8 %
Accepted Summary
0 25 50 75 100 125 150 175 200 225 250 275 300 325 350 375 400 425 450 475 500
1.15
1.64
1.83
1.01
1.36
0.73
0.30
1.27
1.14
1.38
1.54
2.03
1.70
1.42
1.28
0.84
1.13
1.34
0.84
40' Av 1.45% Carb
80' Av 1.35% Carb
80' Av 1.35% Carb
80' Av 1.35% Carb
145' Av 1.75% Sul
225' Av 1.53% Sul
230' Av 1.53% Sul
265' Av 1.44% Sul
0-80 ft
80 ft Carb Ore
Av. 1.35% Cu
80-310 ft
230 ft. Sul Ore
Av. 1.53% Cu
Coordinates Z-100
Elevation of Surface 2030.69
Elevation of Collar 2020.78
Elevation of Bottom 1675.53 ft.

Hole X
Coordinates Z-100

Diamond Drill Samples								Description of Rock	Date	Length of Core		Remarks
Combined	Core	Sludge	Core Composite									
% Cu	% Cu	% Cu	SiO_2 %	Fe %	Al_2O_3 %	CaO %	S %					
1.52 2.02 1.60 1.55 1.49	1.48 1.50 1.73 2.06	1.61 1.53 1.47	82.1 "	4.6 "	4.9 "	0.8 "	0.3 "	Coarse Sil Mon Mal Seams Little Fe Stain	8-19-12 8-20-12	10'–15' 15'–20' 20'–25'	15" 44" 9"	E Bit Medium hard and broken " " " " " " " "
1.66 1.39 1.55 1.07 0.97	0.90 1.41 2.03 1.09 0.59 0.76	1.70 1.33 1.59 1.09 1.00	"	"	"	"	"	More Fe Stain Little Mal. Traces of Diss Cp	8-21-12	25'–30' 30'–35' 35'–40' 40'–44' 44'–50'	27" 18" 16" 9" 24"	Hard and broken " " " " " " " " " " " "
0.89 1.19 0.95 1.34 1.45	0.93 1.61 0.92 0.92 1.24 1.24 1.33	0.88 0.95 1.49 1.45 1.46	"	"	"	"	"	Hard Sil Little Fe and Cu Stain More Fe Stain More Mal on Seams	8-23-12 8-24-12 8-26-12	50'–55' 55'–60' 60'–65' 65'–70' 70'–73' 73'–75'	23" 25" 6" 27" 23" 11"	Medium hard, very broken Cemented at 55' Medium hard and broken Lost water & cemented at 60' Soft & broken Hard & broken Lost Water, cem at 73' Medium hard, badly broken
0.94 1.35 1.00 1.21 1.98	0.87 1.27 1.08 1.11 1.36 2.20	0.94 1.44 1.35 1.00 1.20 1.96	" 81.9 "	" 5.2 "	" 4.5 "	" 1.3 "	" 1.25 "	Softer. More Fe Stain Soft. Diss Cp	8-27-12 8-28-12	75'–79' 79'–81' 81'–85' 85'–90' 90'–95' 95'–100'	5" 12" 14" 14" 17" 18"	Slightly more solid, Cemented at 81' Medium hard broken {Drilled through 60' of Cement Hard and broken " " "
2.62 1.91 3.15 1.59 2.08	1.50 2.21 2.58 1.40 1.87	2.72 1.88 3.16 1.60 2.08	"	"	"	"	"	Hard, Sil Cp Diss & on seams Highly silicified Mon. Cp Diss and on thin seams	8-30-12	100'–105' 105'–110' 110'–115' 115'–120' 120'–125'	18" 23" 4" 7" 10"	" " " " " " Medium hard, broken, cemented at 117½' Soft badly broken, drilled out 30' of cement
2.46 2.10 1.55 2.12 1.82	0.93 1.31 1.93 1.38 1.24 0.41	2.40 2.10 1.55 2.12 1.82	"	"	"	"	"	Much Diss Cp Highly Sil " " " " "		125'–130' 130'–135' 135'–140' 140'–145' 145'–153'	3" 3" 14" 8" 20"	" " " " " " Medium hard, very broken " " " " " " " "
1.78 1.66 1.26 1.24 1.59	1.69 1.00 1.01 1.93 0.86 1.88	1.78 1.66 1.26 1.24 1.59	"	"	"	"	"	Less Qtz & Cp Much more Qtz. Much Qtz. Little Cp	8-31-12	153'–155' 155'–160' 160'–165' 165'–171' 171'–175'	6" 12" 31" 29" 15"	Hard and broken " " " " " " Medium hard and broken " " " "
2.16 1.78 1.13 1.33 1.63	1.23 1.36 1.27 1.55 2.80	2.16 1.78 1.09 1.27 1.46	"	"	"	"	"	Small Cp. Seams Much Qtz, Less Cp Cp increases	9-2-12	175'–180' 180'–185' 185'–190' 190'–195' 195'–200'	14" 12" 39" 48" 27"	" " " " " " " " Hard and broken " " " " " "
1.73 2.13 1.20 1.26 2.06	2.18 1.89 1.65 1.68 1.50 1.10 1.34 [illegible]	1.62 2.22 1.16 1.25 2.08	"	"	"	"	"	Much Qtz Cp diss Cp Seams	9-3-12	200'–205' 205'–210' 210'–216' 216'–218' 218'–225'	48" 37" 66" 14" 42"	" " " " " " " " " " " " " " "
1.08 0.88 0.45 0.61 0.95	1.09 0.60 0.31 0.42 0.66 1.15	1.08 0.89 0.45 0.61 0.68	79.9 "	3.8 "	7.0 "	2.3 "	0.9 "	Broken, Altered, Less Qtz & Cp Hard Sil, Coarse Mon Little Min.	9-4-12	225'–230' 230'–235' 235'–240' 240'–245' 245'–250'	11" 12" 15" 13" 5"	" " " " " " " " " " " " " " "
1.32 0.92 1.19 0.85 1.35	0.87 1.00 1.14 1.03 2.52	1.34 0.92 1.20 0.84 1.18	"	"	— "	"	"	Little diss Cp Slightly more Cp Broken, finer grained, Sil Cp bunches		250'–255' 255'–260' 260'–265' 265'–270' 270'–275'	8" 11" 27" 12" 30"	Medium hard and broken " " " " Hard and broken
1.09 1.41 1.68 1.50 1.22	1.32 1.08 1.89 1.35 1.39 0.76	1.06 1.49 1.66 1.53 1.24	"	"	"	"	"	Light colored Mon Bunches of Qtz with Cp and little B Slips with blue gouge Narrow Cp Seams	9-5-12	275'–280' 280'–285' 285'–289' 289'–293' 293'–300'	24" 36" 19" 36" 21"	" " " " " " Changed to light bit " " " " " "
1.74 1.08 0.69 0.90 0.78	0.76 1.31 0.58 0.80 0.87 0.80	1.83 1.06 0.70 0.90 0.78	"	"	"	"	"	Rather dark Por-Little min only on narrow seams Light, coarse Mon again, Cp diss. and on seams Much Qtz. and Cp	9-6-12	300'–305' 305'–310' 310'–315' 315'–320' 320'–325'	14" 18" 45" 30" 38"	Medium hard and broken " " " " Hard and broken " " "
1.22 1.06 1.27 0.90	1.18 1.44 1.72 0.66	1.24 1.00 1.21 0.92	"	"	"	"	"	Less Cp Sil Mon Diss Cp		325'–330' 330'–335' 335'–340' 340'–345'	38" 26" 27" 18"	" " " " " " " " " Hole stopped at 345.3 ft

Carbonate Mineralization
Sulphide Mineralization

- Mal – Malachite
- Az – Azurite
- Ccc – Chalcocite
- Cp. – Chalcopyrite
- B – Bornite
- P – Pyrite
- Qtz – Quartz
- Mon – Monzonite Porphyry
- Rhy – Rhyolite
- Diss – Disseminated

DIAMOND DRILL HOLE No. X
AJO EXPLORATIONS

the core assay, it is best to split every piece of core,[3] sending one half of it to the assay office and retaining the other half in a marked box or bag for record. It is convenient to keep the core in oblong tin boxes, marked on one end, in racks, so that any box is easily accessible. A portion of the sludge, returned from the assay office, should be kept with the core.

The third requisite is to combine core and sludge assays for every sampling interval in such a proportion as to give the assay content of all material removed from the hole. This is done by measuring the length of core obtained in every sampling interval, computing the volume of core and the volume of the rest of the material removed from the hole (which has been ground into sludge), and combining core and sludge assays in proportion to the volumes. This work may be simplified by formulas,[4] or by a very ingenious scale devised by the E. J. Longyear Co., on which the proportional weight to give to core and sludge assays for any size of bit is read directly opposite a scale showing lengths of core recovered in 5 ft. of drilling.

The drillmen and samplers should submit daily reports for every hole, giving the progress and detailed information concerning the material cut, whether or not the ground is caving, the length of core for every 5 ft. advance, and all other factors that might affect the accuracy of the samples. When a hole is finished, a chart should be prepared giving core, sludge, and accepted assay, with a graphic representation of the accepted assay, and columns showing material cut, minerals present, dates, causes of delay, factors affecting the accuracy of sample, and any other information of interest. These charts are the permanent record of the drilling; a specimen chart for one of the Ajo drill holes is given in Fig. 1.

In diamond drilling, the core assays are usually high if the values are in hard, siliceous bands, as often at Ajo, and low if in seams softer or more friable than the gangue. Sludge assays are low in the first case, high in the second. The combined assay should represent closely the average value of the rock drilled. Except in loose ground, there is little jarring down of material from the walls of the hole. It is only where the values are in very soft seams that the sludge and combined assay may be materially salted by richer material from the walls. After a few check raises or test pits have proved that there is not much of this salting, diamond-drill samples can be relied on to within the accuracy of ordinary assaying.

It is interesting to note that a little oil is carried down the hole from the drill, thoroughly churned with the sludge, and on coming out of the

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[3] Hugh M. Roberts and R. D. Longyear: Genesis of Sudbury Nickel-copper Ores *Trans.* (1918) **59**, 42.

[4] E. E. White: *Op. cit.*

hole, floats away as a thin scum, carrying with it small particles of sulfides. Microscopic examination of this scum at Ajo showed quite appreciable amounts of sulfides floating with the oil on top of sludge barrels. This floating away of sulfides may give a small factor of safety to the sludge assays.

Churn-drill Sampling

Sampling of churn drill holes has been well described by E. R. Rice.[5] As in the case of diamond drilling, it is important to clean out the hole thoroughly at the beginning and end of every sampling interval. The process of sampling is simple, as the sludge brought up by the bailer runs through a mechanical splitter which automatically cuts it to the proper size—usually a wash tub full. It is important to wash the launder and splitter at the end of every sample, and to include a proper proportion of the wash water in the sample. As the amount of water in the sample is not excessive, it is better to evaporate off all the water instead of allowing the sludge to settle and siphoning or pouring off the water. Mr. Rice suggests drying by steam instead of coal or wood, to prevent roasting the sulfides in the samples.

As in the case of diamond-drill holes, detailed daily reports should be made for every hole; and when a hole is finished, a chart should be prepared for the permanent record, giving a graphic representation of the assay, and all features of possible interest noted during the drilling. Mr. Rice gives forms for daily reports and charts. A part of the sludge from every drilling interval, quartered out before grinding, should be kept for reference, preferably in a tin box.

Churn drilling is somewhat less accurate, in most cases, than diamond drilling. Unless the hole is cased nearly to the bottom, the frequent passage up and down of the bit and bailer knocks off material from above the sample. If the values are in soft seams, the sample will be too high grade. Even if the ore minerals are disseminated through the rock, the sample will be affected by material from higher up in the hole. As Mr. Rice suggests, if the hole is kept filled with water to above the bottom of the casing, the water pressure tends to hold material in the walls in place. The inaccuracy of churn-drill samples is lessened by the comparatively large size of the hole. A 5-ft. advance of an 8-in. (20 cm.) hole cuts out about 200 lb. (90.7 kg.) of ordinary disseminated ore. If this ore assays 1.5 per cent. copper, and if 5 lb. (2.3 kg.) of 5 per cent. ore are shaken down from crevices in the walls of the hole, salting the sample, the resulting assay would be 1.58 per cent. It is seldom that a sample would be salted more than this.

With careful work, involving casing below all caving ground, assays

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[5] Churn Drilling of Disseminated Copper Deposits. *Eng. & Min. Jnl.* (June 25, July 2, 1921).

from churn drilling can be depended on in nearly all cases to within 0.1 per cent. copper. In every property, a large amount of raising or test-pitting should be done to check the drilling. If drilling results are high, a factor of safety is thus found to use in future drilling at that property.

Sampling of Underground Workings

The best sample from underground workings is made up of all the material removed after every round is shot. On account of the great weight of the sample, the salting by rich ore falling in from friable seams in the rock is not considerable. If, as an opening advances, every tenth shovel or bucket full is thrown into a sample container, and the samples thus obtained are properly crushed and quartered—preferably by a mechanical sampler—the results are as nearly fool-proof as sampling can be. The only possible error comes from the strange impulse that moves the most ignorant and disinterested workman to throw the rich chunks into the sample. At Ajo, the large test-pit samples, made up of every tenth bucket full of ore hoisted, averaged 0.15 per cent. copper higher than corresponding channel samples. This error can be accounted for only by the fact that Mexicans have an inherited adeptness at sorting ore, which unconsciously influenced them in filling the tenth bucket. This example illustrates the fact that no sample can be blindly trusted.

Grab samples from rounds shot in underground workings are almost always high. This is partly because of the tendency to take too great a proportion of fines, and partly because of the almost magnetic attraction of the most conscientious hand toward rich pieces of ore. Grab samples should be used only as rough checks on the other methods of sampling.

Channel samples must be taken where existing underground workings must be sampled, or where it is not practicable to take a proportionate part of every round as it is mucked. The accuracy of this method of sampling depends on laying down a rigid system of sampling, and adhering to it. The location and direction of channels should be most carefully studied, and the same channel interval should be used throughout the orebody. If the mineralized seams tend to follow any prevailing structure, the channels should cross this structure. Usually these seams run in all directions, and it is safer to cut all channels in the same direction—horizontally or vertically on one or both walls—throughout the orebody. Some of these channels may follow rich seams, but the average should be about right. The channel interval chosen is usually 5 or 10 feet.

It is most important to clean off the rock before channeling. The best way is to use a pick and then a stiff brush. This cleaning removes the dust that has collected on the rock, and also tends to rub or jar out a little of the richer ore from seams before sampling begins. This loss

partly compensates for the excess of rich material that falls into the sample from seams in the bottom of the channel.

The size of channel depends on the uniformity of the ore. If this is hard and uniform, channels 4 in. (10 cm.) wide by 1 in. (2.5 cm.) deep are sufficient. Often, even small samples taken with a hand pick give fair results. In very blocky and irregular ore, in which a large part of the mineral is on seams, large channels, up to 6 in. wide and 3 in. deep, should be cut. In such uneven ore, it is useful to provide samplers with wooden blocks the width and depth of the required cut, and to make these blocks fit in the channel at all points. With these blocks, experienced samplers cut very uniform channels even in badly jointed ore.

Where channels are small, the samples may be caught in boxes or bags. Where large, it is much better to spread large canvasses in the bottom of drifts, or on platforms in raises and test pits, and to cut the samples on them. One engineer can watch two or three crews of samplers, working on adjacent cuts, and make sure that the channels are evenly cut and that any rock from outside the channel which falls on the canvas is thrown to one side.

If crushers are available, the entire sample should be sacked, tagged, sent to the crusher, and then put through a mechanical splitter. Anything save a preliminary rough sampling of a disseminated property would justify the purchase of this equipment. If the equipment is not available, the sample may be broken to ½ in. or so by hand, and quartered, with finer hand crushing between quarterings, to a 4 or 6-lb. sample, which is sent to the assayer.

The results of all sampling should be recorded on assay maps, usually on a scale of from 50 to 20 ft. (15 to 6 m.) to the inch (2.5 cm.). Entering the assay, instead of the number of sample, on the map simplifies the later work of combining assays and estimating.

If the channels are sufficiently large for the character of ore, channel samples can be accepted with more confidence than either churn-drill or diamond-drill samples. Channel samples come from the desired place, while drill samples may be salted by ore from an upper part of the hole. Morton Webber[6] has pointed out that there are certain errors latent in hand sampling. If the valuable minerals are harder than the gangue, the samples tend to be low; and if softer or more friable, high. These latent errors are less important in copper than in gold and silver ores. The mineral itself is less valuable, and a far greater quantity of it is required to affect the sample. A larger channel can be cut in fairly soft disseminated ore than in hard gold quartz, so that salting, by friable ore minerals, from the sides and bottom of the channel has less effect. Also the cleaning of the rock before channeling makes up for some of the ore that falls

[6] The Combination Method of Hand Sampling. *Min. & Sci. Pr.* (Feb. 28, 1920).

in from the bottom of the channel. If a measuring block is used, there is less chance of cutting too much of softer bands, and too little of harder bands in the ore. If there are workings enough to require several hundred samples, minimizing the effect of a few high-grade assays, channel sampling should be accurate to within 0.05 per cent. copper.

Checking Sampling

The ideal method for determining the assay content of a disseminated orebody is to check at least every fifth drill hole by a raise or test pit 50 or 100 ft. (15 or 30.5 m.). high or deep. One tenth of all material removed from each round shot in these workings should be taken as a bulk sample, and the walls should be carefully channeled. As a further check, drifts should be driven between several test pits or raises, and bulk channel samples taken in them. By this combination method, the value of the deposit is absolutely proved, and the results of future drilling, raising, or test pitting can be trusted without further checking.

As an example of results of the three methods of sampling, in the development of the New Cornelia Copper Co., at Ajo, Ariz., over 1000 ft. (304 m.) of test pitting was done to check diamond drilling. The average of diamond-drill samples and of large channel samples checked to within 0.005 per cent. copper. The bulk samples, consisting of every tenth bucketful crushed and quartered mechanically, averaged 0.15 per cent. higher than the channel and drill samples. The drill and channel samples were accepted as correct. Several million tons of this ore have now been mined and treated. All of this ore to date has averaged 1.51 per cent. copper, compared with the estimate of 1.54 per cent. The grade has been a little higher than the estimate in one part of the orebody and lower in another part. This property has proved that, in a disseminated orebody, it is possible to sample and estimate the ore correctly without reducing the grade by a factor of safety.

Estimation of Ore Reserves

In nearly all of the disseminated copper orebodies, in the opinion of the writer, the sampling has been well done and has given accurate results. Yet in many of the properties the ore mined has averaged from 0.1 to 0.4 per cent. copper lower grade than the estimates. This is due to an error in the method of combining assays to give an average grade and length of ore in every drill hole or other opening, and to a failure to base the estimate on mining conditions. These are the difficult problems in the estimation of ore reserves. A knowledge of the geology of the deposit is necessary for the interpretation of changes in the grade of ore, and for the proper combining of assays to give the average grade. Also, a fair understanding of the size and shape of the deposit and of the character of ore is necessary

for the determination of the methods of mining and of treating the ore, which in turn govern the minimum grade that can be mined with profit. The mining method and the minimum grade again modify the shape and grade of ore that will be mined. All of these factors are interdependent. Without thorough study of them, and particularly of the geology of ore occurrence, care in sampling is thrown away.

Use of Geology in Ore Estimation

The general form of nearly all disseminated copper orebodies is tabular; that is, the horizontal dimensions are much greater than the vertical. But the fractures along which mineralization is strongest nearly always dip steeply. In the main part of most orebodies, the mineralization is so intense that the rock between the fractures is commermercial ore. In outlying portions, both around the edges and underneath the main orebody, the rock between fractures is less thoroughly mineralized and is usually below the minimum grade, while along the fractures there are bands of richer ore, often several feet wide. These richer and leaner bands are too narrow to be mined separately without great expense. If rich and lean together average above the minimum, the whole mass will be mined; if below it, all will be left.

This banded character of outlying ore is exaggerated in drill holes or other vertical openings. Fifty feet of 2 per cent. ore in a drill hole may be caused by a 5-ft. band along a steeply dipping fracture. The next 50 ft. of 0.5 per cent. material may represent only a 5-ft. band of less thoroughly mineralized rock between fractures. If the hole is estimated to have cut 50 ft. of 2 per cent. ore, the mining grade will be far under the estimate. It should properly be figured to have cut 100 ft. of 1.25 per cent. ore, assuming that the 2 per cent. band has persisted to 100-ft. depth.

Around the edges of an orebody, even with the most careful study of ore occurrence and fractures, it is often impossible to determine just how much mineable ore a drill hole represents. A few feet to one side, the conditions may be absolutely different from those in the drill hole, depending on the regularity and frequency of mineralized fractures. For this reason, it is good practice to figure "developed ore" only to the outlying drill holes, not beyond them. Even with this limitation, estimates of outlying portions of orebodies often do not include enough of the lean bands and give too high a grade. In determining the probable bottom of commercial ore, under the main orebody, a closer approximation can be made. Where the material from the bottom of the main stretch of ore cut in a drill hole to the bottom of a lower band of ore, including rich and lean bands, averages above the minimum, it is safe to include this whole length and grade in the orebody. But if the average of lean and rich bands is below the minimum, the whole should be thrown out. Even where the lean and rich bands below the main orebody are 50 to 100 ft. thick, it is

almost never safe to estimate that leaner material can be left and the deeper ore mined. This assumption should only be made when there is definite geological evidence that a lower orebody of commercial size exists.

Fig. 2 is a typical example of a section through four drill holes that cut a lean capping, then a body of good ore, then a band of material of lower grade than the 1 per cent. minimum, and finally a lower band of ore. The average assays are given in the section. The temptation is to estimate two nearly parallel horizontal orebodies, as shown by the dotted lines. But the fractures are steep, and there is no geological structure to cause a lower horizontal orebody. Probably the alternate lean and richer

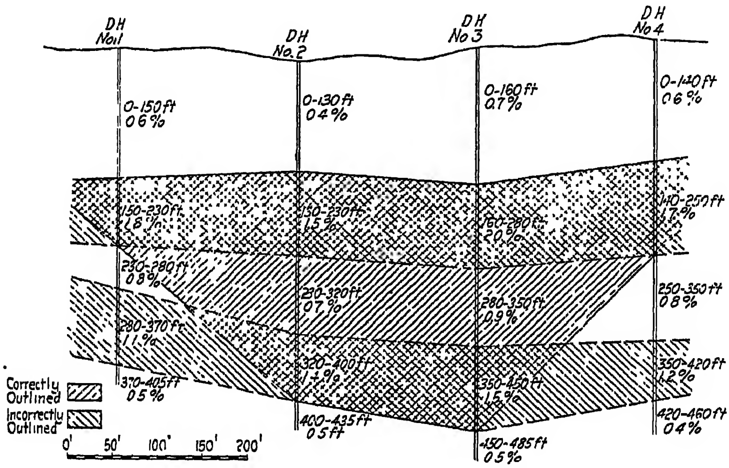


FIG. 2.—VERTICAL SECTION THROUGH PART OF DISSEMINATED OREBODY, SHOWING CORRECT AND INCORRECT OUTLINES OF ORE DEDUCED FROM DRILL HOLES.

bands are caused by deep enrichment along fractures and unenriched material between them. The apparent thickness of the richer bands is caused by the steepness of the enriched fractures. Either all the material below the main part of the orebody must be mined at any point, or none of it. Averaging lean and rich bands, in proportion to their lengths, gives the following results:

Drill hole No. 1, 230 to 370 ft., 0.99 per cent.
Drill hole No. 2, 230 to 400 ft., 1.03 per cent.
Drill hole No. 3, 280 to 450 ft., 1.25 per cent.
Drill hole No. 4, 250 to 420 ft., 0.97 per cent.

With the 1 per cent. minimum, the material below the main part of the orebody in holes No. 1 and No. 4 is below the commercial grade and must be rejected. That in holes No. 2 and No. 3 is above the minimum and will be mined.

The depth and average grade of ore estimated in these four holes should be:

Drill hole No. 1, 80 ft., 1.80 per cent.
Drill hole No. 2, 270 ft., 1.20 per cent.
Drill hole No. 3, 290 ft., 1.56 per cent.
Drill hole No. 4, 110 ft., 1.70 per cent.

If for comparison each of these holes is assumed to represent a block 200 ft. square, with 12.5 cu. ft. to the ton, the holes have developed 1,800,000 tons of ore averaging 1.47 per cent. copper. The incorrect method of estimating, assuming two layers of ore separated by waste, gives 1,800,000 tons averaging 1.56 per cent. copper. Mining would probably check the first estimate and grade very closely. This case is typical. The wrong method of combining assays rarely involves a great error in tonnage, as the lean material left out about compensates for richer material included. But the grade given by the incorrect method is considerably higher than the deposit will mine; and a mining plan based on it will not be suited to the form of the deposit. Inspection of drill-hole records of several disseminated copper properties indicates that this error in combining drill-hole assays, caused by lack of understanding of the ore occurrence, is by far the most important reason for the failure of many properties to mine up to the estimated grade.

The other fact emphasized by geological study is the great variation of copper content in most orebodies of this type. A drill hole seldom really represents the ground for 100 ft. on every side of it. Close checks in small areas cannot be expected. It is therefore necessary to have a great number of drill holes, spaced regularly, not farther than 200 ft. apart. If this is done, in the whole orebody the average ore indicated by drilling will check closely with the average of ore mined.

Study of Mining and Costs before Estimating

Before an estimate can be made which will check the mining, it is necessary to know what mining method will be used, and what the total cost of producing copper, and so the minimum grade of commercial ore, will be. If steam-shovel mining is selected, in order to place the approach properly, it will be advantageous to mine and treat some material of lower grade than the minimum, and to leave in the sides of the pit some ore above the minimum grade. If a caving system is chosen, allowance must be made for mixture of ore with capping. This mixture will result in the loss of part of the ore and in the sending of some lean capping material to the mill. The estimate must be modified to allow for this mixture; also the sending of waste to the mill will increase the charges against the ore, and so affect the minimum grade and the tonnage.

C. E. Arnold has discussed the effect of mining method on cost and extraction.[7] The importance of the effect of a mining method on an estimate is shown by the fact that in one of the disseminated properties in which a caving system was used the manager stated that he was mining 105 per cent. of the estimated tonnage and 90 per cent. of the estimated copper. This estimate should have been recalculated.

The tonnage, grade, and form of an orebody must be known in order to determine the mining system, costs, and minimum grade of ore that can be mined; and the mining system and minimum grade must be known in order to estimate the grade and tonnage of ore that will be mined. All the factors are interdependent. The best solution to the problem is to make a preliminary estimate, based on an assumed minimum, and neglecting any influence of mining method on the outline of ore mined. This estimate will give the approximate form, grade, and tonnage in the orebody. From it the mining method, method of treatment, and approximate costs may be determined, and a more accurate minimum grade calculated. Using this new minimum, a final estimate is made, making the proper allowance for the effect of the mining method chosen on form of orebody and cleanness of mining. This final estimate should agree almost exactly with the tonnage and grade that will be mined. The only material difference will be caused by improved operating methods, process, or recovery, which will make it possible to lower the minimum grade of profitable ore and so to increase the tonnage.

Calculation of Tonnage

The calculation of tonnage is a simple matter. Where the deposit is developed by drill holes on coördinates, the average depth of ore in holes on the four corners of every square, multiplied by the area of the square, gives a sufficiently close approximation to the volume of ore in the square block. The sum of the products of assay times depth of ore in the four holes divided by the sum of the depths of ore gives the average grade of ore in the block. If drill holes are not on coördinates, the deposit is usually divided into triangular vertical prisms, with a drill hole at every corner. The volume and grade of ore in every prism are found just as in the case of the square prisms described. As pointed out by J. E. Harding,[8] this method is not accurate when the triangles are not equilateral, but averaged over the whole deposit, the approximation is close enough unless drill holes have been purposely placed in richer parts of the ore. Whether the deposit is divided into square or triangular prisms, the tonnage of ore in the whole orebody is found by adding the tonnages in all the prisms, and

[7] Cost and Extraction in the Selection of a Mining Method. *Trans.* (1916) **55**, 203.

[8] Calculation of Ore Tonnage and Grade from Drill-hole Samples, 39. Paper No. 1035 with *Min & Met.* (Dec., 1920).

the grade of ore in the deposit is found by dividing the sum of the products of tonnage and grade in all the prisms by the total tonnage.

In order to reduce cubic feet of ore in the various blocks to tons, specific gravity tests should be made on samples of coarse and fine ore from workings in various parts of the deposit. By weighing the samples in and out of water, subtracting the weight of the container in each case, the weight of ore per cubic foot is found by the formula:

$$\text{Weight per cubic foot} = 62.5 \times \frac{\text{Weight of ore in air}}{\text{Weight in air minus weight in water}}$$

Disseminated copper ore averages about 12.5 cu. ft. per ton. The average factor found in any deposit is used for that whole orebody.

The final estimate, based on the calculated minimum and the plan of mining adopted, is often made by passing parallel vertical sections through the orebody along coördinate lines. These sections show all the ore recoverable by the method chosen. The areas of sections of the orebody thus outlined are measured, and the average grade of ore in each section is obtained by dividing the sum of the products of grade times depth of ore in all drill holes in the section by the sum of depths of ore. The volume between adjacent sections is computed by multiplying the average between the areas of the sections by the distance between them, and the grade of this ore is the average in proportion to areas between the grades in the two sections. If one section is much smaller than the next, the prismoidal formula may be used. (Volume of prismoid equals distance between sections multiplied by one-sixth the sum of the two end areas plus four times the area of the section midway between them.) It is seldom necessary to use this formula. If the drill holes are not on coördinates, it is simpler and sufficiently accurate to make the final estimate like the first by dividing the orebody into triangular vertical prisms, making the areas and shapes of the prisms conform with the outlines of ore that will be mined.

If the deposit is developed partly by vertical and partly by horizontal workings, the method of estimating must be decided by careful study of the individual case. The orebody is usually divided into simple geometrical figures, with workings along as many of the edges as possible. Great care must be taken not to give undue weight to assays in drifts that follow the mineralized structure; it must be remembered that workings on one level may be at the horizon of greatest enrichment. Assays from them must not be given too much weight compared with those from raises or winzes. In general, samples from workings on coördinate lines are much more reliable than those where the plan of workings is not regular, as, in the latter case, it is almost certain that the richer portions of the orebody have more than their share of workings. An estimate in such a case depends entirely on the judgment of the engineer.

If there is likely to be a mixture of ore with capping, the final estimate should be modified by factors that allow for the loss and dilution of ore and for the resulting lowering of grade.

Summary

The sampling of a disseminated copper orebody is simple and should be very accurate. The estimation of tonnage and grade of ore in the deposit, once the length and grade of ore in every opening has been figured, is a simple mathematical problem. There is no excuse for great errors except where the irregularity of workings suggests the probability that richer portions of the orebody have been more thoroughly developed. The combination of individual assays in a drill hole or other opening to give the length and average grade of ore that will be mined is the difficult part of the problem of estimation. To solve it, the engineer must have an intimate knowledge of the geological occurrence of the ore, of the mining method to be used with its effect on recovery of tonnage and of copper, and of the total probable costs per ton of ore and per pound of copper, indicating the minimum grade of ore that will be treated at a profit. The failure to study these factors properly is responsible for the fact that the ore mined in many disseminated copper properties has averaged from 0.1 to 0.4 per cent. copper lower than the estimates. This is an unnecessary error. The final estimate should check the tonnage mined almost exactly, and the grade of ore mined within less than 0.1 per cent. copper.

Sampling and Estimating Orebodies in the Warren District, Arizona

BY ROBERT H. DICKSON,* BISBEE, ARIZ.

(San Francisco Meeting, September, 1922)

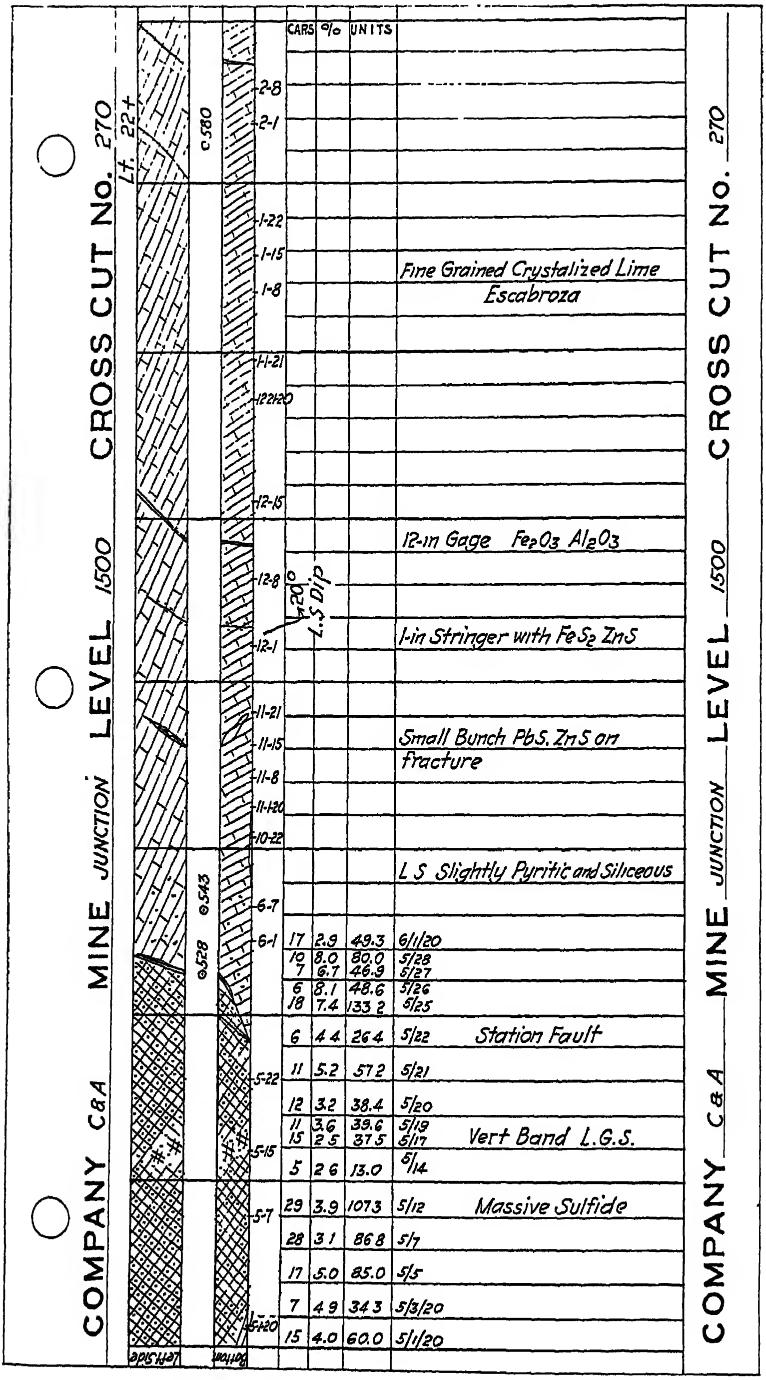
LIMESTONE replacement and contact metamorphic orebodies are so irregular that sampling and estimating are far less exact than in more regular types of deposits; both the mineralized masses and the lenses of ore within them are extremely irregular. So much development work is required to find the ore that the expense of blocking out great tonnages is prohibitive; furthermore, the rock near these orebodies is so heavy that drifts or raises cannot be kept open for many years without excessive repairs. As a result, few mines in limestone have more than three years supply of ore in sight. As most of the ore is developed by stoping—not by drifting, raising, or drilling—estimates of ore developed do not indicate the total amount of ore that may be expected; they simply indicate the relative condition of the mines. But even if they are to do this, they must be made according to a fixed plan, intelligently carried out. The estimates of ore in the Calumet & Arizona Copper mines, Warren District, Ariz., have been of much value to those who plan the mining and development work. This paper describes the methods of sampling and estimating used in these mines.

SAMPLING

Samples are taken in various ways and depend on the condition of the working and the purpose for which the sample is to be used. Drifts, crosscuts, and raises are sampled by channels or by taking a grab off each mine car as it is filled. Stopes are usually sampled by drill holes or by taking a representative sample off each mine car; the latter, or chute, sample is by far the best sample.

All workings are studied periodically by an engineer, who makes graphic and detail notes of the geologic structure, as shown in Fig. 1. The daily car tally and assay of each particular working is added to the same sheet opposite the material it represents. These sheets make a permanent record, showing the exact boundaries as well as the grade of ore in all workings.

* Chief Engineer, Calumet and Arizona Mining Co.


COMPANY C&A MINE JUNCTION LEVEL 1500 CROSS CUT No. 270
Lt. 22+
c 580
c 528
c 543
2-8
2-1
1-22
1-15
1-8
1-1-21
12-21-20
12-15
12-8
12-1
11-21
11-15
11-8
11-1-20
10-22
6-7
6-1
5-22
5-15
5-7
5-1-20
CARS %o UNITS
Fine Grained Crystalized Lime
Escabroza
12-in Gage Fe_2O_3 Al_2O_3
20°
L.S Dip
1-in Stringer with FeS_2 ZnS
Small Bunch PbS. ZnS on fracture
L S Slightly Pyritic and Siliceous
17 2.9 49.3 6/1/20
10 8.0 80.0 5/28
7 6.7 46.9 5/27
6 8.1 48.6 5/26
18 7.4 133 2 5/25
6 4 4 26 4 5/22 Station Fault
11 5.2 57 2 5/21
12 3.2 38.4 5/20
11 3.6 39.6 5/19
15 2 5 37 5 5/17 Vert Band L.G.S.
5 2 6 13.0 5/14
29 3.9 107 3 5/12 Massive Sulfide
28 3 1 86 8 5/7
17 5.0 85.0 5/5
7 4 9 34 3 5/3/20
15 4.0 60.0 5/1/20
Left Side
Bottom
COMPANY C&A MINE JUNCTION LEVEL 1500 CROSS CUT No. 270

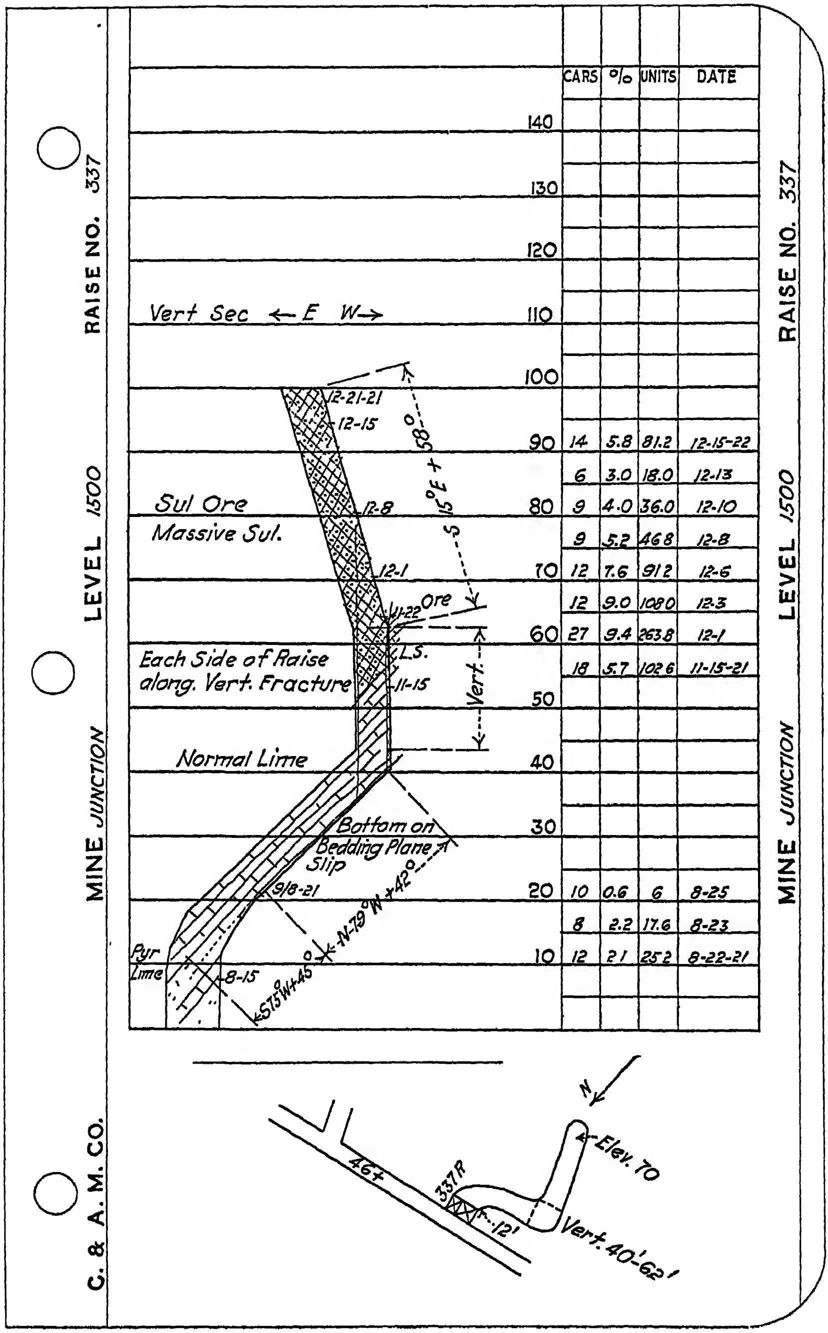


FIG. 1.—GEOLOGIC AND ASSAY MAP OF WORKINGS.

Mapping

The geologic data are transferred to a set of maps (tracings) drawn on a scale of 1 in. to 50 ft. These stope maps are horizontal sections, on which all workings are plotted, taken 8 ft. apart starting at each level. Samples are plotted, when necessary, and various classes of ore are shown by symbols. These horizontal sections, supplemented by a series of vertical sections, taken at right angles to the general direction of fissuring, furnish the base for outlining ore estimates (see Fig. 2).

Estimating

Estimates of developed ore are calculated semiannually, while those of probable and possible ore are made when of special interest. The estimate of developed ore furnishes the operator with a good idea of the actual tonnage and grade of ore in the various stoping areas. The probable ore estimate indicates the probable life of certain areas or orebodies that have been partly developed, while the estimate of possible ore affords an approximate idea of the life of the mine.

Developed Ore

The question of what can be considered "developed ore," or "ore in sight," varies with conditions, stage of mining operation, and ideas of the estimator. As usually considered, it means ore blocked out in three dimensions, allowing an arbitrary footage of ore beyond the faces exposed. Ore occurring persistently in a large lens of fairly uniform grade with regular boundaries, developed on levels 100 ft. apart, might be considered developed ore; whereas, where the orebodies are known to be small and erratic, the developed ore might be considered only as that ore within the probable shape of the lens and extending not over 10 ft. from a known face of ore.

During the initial development stage, greater latitude can be taken in considering ore developed than is possible in later stages. In the case of large uniform orebodies, ore partly blocked out on two successive levels with enough raises between to show the continuity of same, is considered developed ore in the early development stage. At present, however, only small parts of the orebodiess are regular, uniform in grade, or large enough to be considered in this class. The general system of estimating developed ore, which applies to most of the smaller, or erratic, lenses and to the larger, more fully developed orebodies, is to outline the probable shape of the orebody and to include within this, as developed ore, all the ore lying within 20 ft. of any known face. In other words, that ground lying within an outline passing 20 ft. outside of known exposures of ore, is considered to be developed ore, unless from a knowledge of the geologic data affecting or bounding the orebody, it is known to be otherwise.

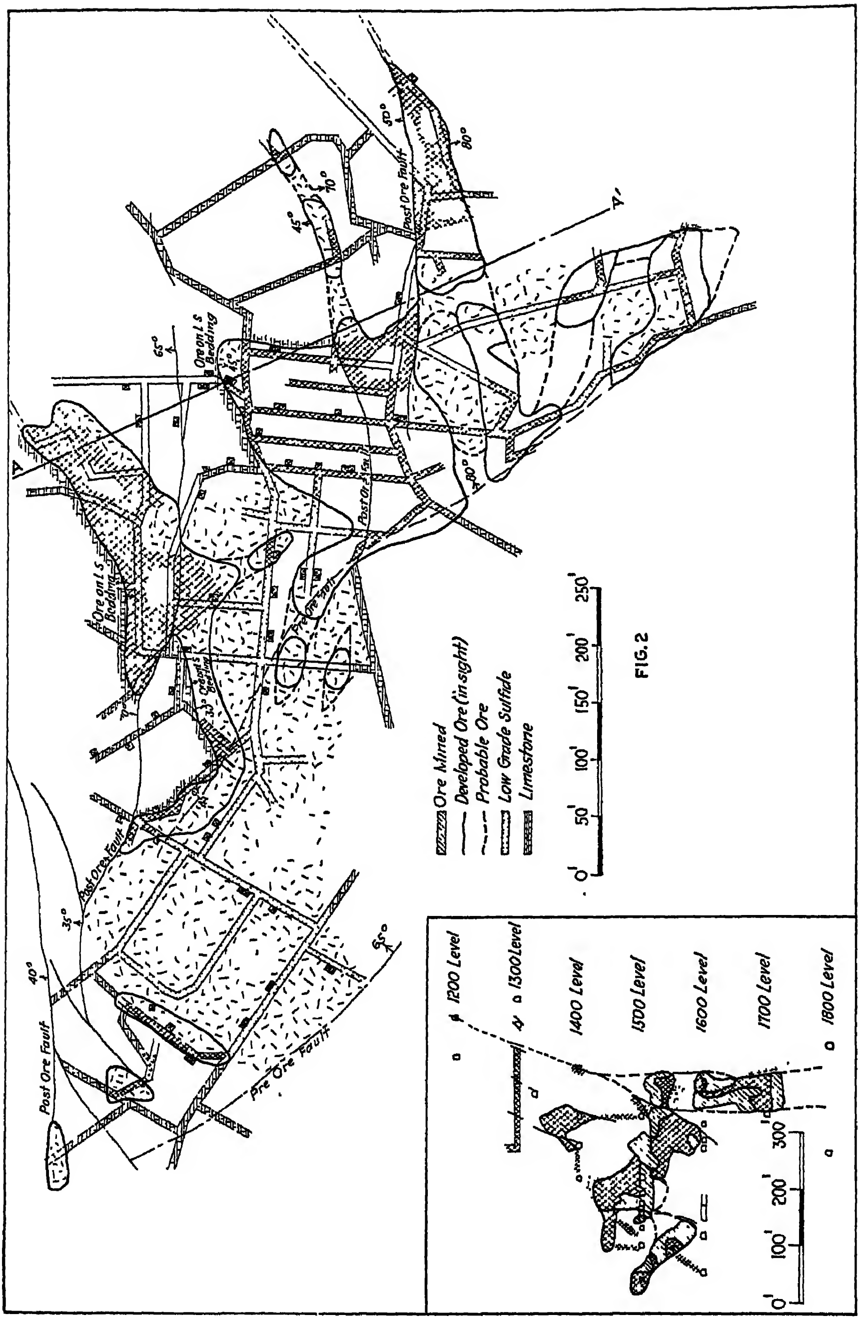


FIG. 2.—DEVELOPED ORE AND PROBABLE ORE ESTIMATES; INSERT IS A VERTICAL SECTION ALONG AA'.

When estimating narrow high-grade veins, streaks, or bunches, the mining method with consequent stoping width is considered. The assay value assigned to all estimates is obtained by combining the assays of the ore cut in enclosed, or bounding workings, giving a weight to each sample corresponding with the proportion to the whole that it represents; always paying due attention to the weight per cubic foot of mineral composing portion of the sample.

Fig. 3 is part of a stope map with the assays added to illustrate the practice of outlining estimates. This particular estimate is based on a 3 per cent. copper minimum. The developed ore areas are planimeterized and volumes calculated by the usual methods. In cases where no work has been done above or below a level, developed ore is considered to

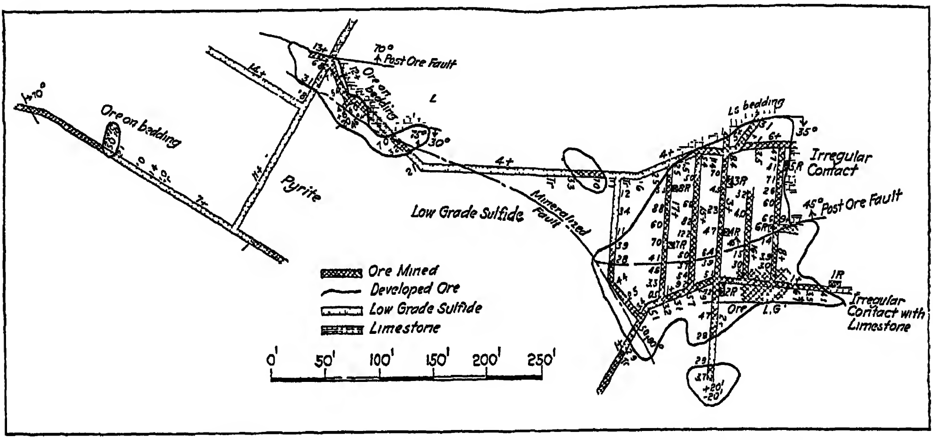


Fig. 3.—Method of drawing "developed ore" estimate lines.

extend to a horizontal plane 20 ft. above or 20 ft. below, unless it is known to be cut off.

For convenience in mining and estimating, the larger ore zones or orebodies are divided into small sections of a fairly uniform grade designated by separate stope numbers. Estimates are first calculated for individual stopes, or series of stopes of uniform grade. These are combined to give an estimate by orebody, level, and mine.

The various classes of ore (as sulfide, oxide, low-grade sulfide, etc.) are segregated in the estimate. In addition to showing the location of tonnages of known ore of certain grades, the estimate of developed ore furnishes a basis of calculating the ore developed (difference of two estimates plus ore mined) in any area and the ratio between ore developed and development work done.

Probable Ore

An estimate of "probable ore" is made on special occasions to determine the probable life of an orebody, or section of the mine. The differ-

ence between this and developed ore is that this estimate includes all the ore lying within the probable shape of the orebody, as determined by known contacts, or by applying a knowledge of ore deposition under similar conditions. As the outline of probable ore can extend a much greater distance from known ore than developed ore, such an estimate is more liberal where the ore-bearing zone is only partly developed; but as the orebody approaches full development, this estimate approaches that of developed ore. The vertical sections play a more important part in outlining the probable shape, than in the case of developed ore.

The "possible ore" in an area is based on the probability of finding the same amount of ore per acre as has been found in other parts of the district under similar geologic conditions. Such an estimate is of considerable aid in assigning a value, or life, to comparatively unexplored areas of a mine or district. It has also proved to be a good approximation in the Warren district, where the ore is scattered under fairly large areas.

Methods of Sampling and Estimating Ore in Underground and Steam-shovel Mines of Copper Queen Branch, Phelps Dodge Corpn.

By R. W. Prouty* and R. T. Green,† Bisbee, Ariz.

(San Francisco Meeting, September, 1922)

The object of this paper is to describe, as concisely as possible, the methods of sampling and ore estimating used at the Copper Queen Branch, Phelps Dodge Corpn., as applied to the Copper Queen mines at Bisbee, Ariz. Because of the difference in occurrence, manner of prospecting, and methods of mining, the sampling and estimating practice falls into two divisions: That for the underground operations, and that for the Sacramento Hill steam-shovel operations.

The orebodies of the Warren district may be divided into three main classes: Replacements of limestone, replacements of porphyry, and replacements of contact breccia. The first class is more or less intimately related to porphyry intrusions, although in badly fractured country this relationship may not be well shown. These orebodies are mainly oxide and carbonate ore and very irregular, both in size and outline, although usually roughly lenticular and following the bedding of the limestones. There is usually quite a sharp boundary between the ore and waste in the carbonate and oxide orebodies and sampling is not necessary to determine the limits of commercial ore. In the sulfide orebodies, it is exceedingly difficult, without sampling, to determine the boundary between commercial ore and the surrounding low-grade or barren pyrite. Calculations made a few years ago showed that these orebodies had an average thickness of 35 ft. As some of the orebodies are from 100 to 300 ft. thick, it can be seen that the smaller ones form a difficult prospecting and estimating problem.

In the replacements in porphyry, both underground and in Sacramento Hill, the boundary usually cannot be determined without sampling. There is a gradual decrease in value until barren material is reached; this gradation may occur within a short distance or it may extend for several yards.

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Surrounding the Sacramento Hill porphyry stock is a wide zone of contact breccia composed of fragments of porphyry and limestone, all intensely silicified. Within this mass have been found many replacements of sulfide, some barren and some carrying enough copper to constitute ore. These deposits are extremely irregular, as to location, size, and copper content, and offer some of the most difficult problems of prospecting. No set of conditions especially favorable to their formation have been recognized and without sampling it is impossible to determine whether a certain mass of sulfide will be commercial ore or not.

Underground Operations

Sampling Methods

Different kinds of samples are taken in the mines, depending on the reason for which the information is desired:

1. *Grab samples*, taken by the shift boss or foreman, are used as a guide in mining operations; these are assayed by the potassium cyanide method and the results sent to the division office as soon as possible.

2. *Chute samples*, taken from the cars as loaded by carmen under shift-boss orders, are used to determine the grade of ore mined in a certain section for the shift or day; these are assayed by the cyanide method and results reported daily.

3. *Belt or shipment samples* are taken by the sampling department from the loading belt at the main hoisting shaft and from the mine cars, when dumped, for the ore that is not hoisted. These samples give production figures for the day and 15-day periods, and are checked against the samples taken at the smelter. These samples, adjusted to conform to those from the smelter, form the base on which calculations are made to show the actual grade mined in each stope for the 15-day period. As separate belt samples are taken for each loading pocket at the shaft and a record kept of the ore passing through each one, a fairly accurate adjustment is possible between the belt samples and those taken at the chutes.

Once a month, a complete analysis is made of a composite of the daily chute samples from each stope. This analysis is used in figuring minimum grades for mining in each section of the mine.

4. *Channel or groove samples*, taken by the sampling department, are used for records of ore passed through in development work and for estimating grade of ore developed before stoping commences. Channel samples are taken in the stopes when difficulty is experienced in determining points at which material below the minimum grade desired is being mined. These samples are taken with a pick, hammer, and moil, or a light air hammer fitted with a moil point. Channels average about 4 by 1 in. and are taken for 5-ft. intervals in uniform ground. Where an abrupt change from ore to waste or especially rich seams are found the

sample is divided and the length of sample in each material noted. In locating channel samples, the common precautions in regard to direction of sample with relation to structure and of cleaning the surface of oxidation products, are observed. Belt and channel samples are assayed by the potassium-iodide method and results reported daily.

In the oxide and carbonate orebodies, little channel sampling is done as the change from ore to waste usually is pronounced enough to be easily recognized by an experienced man. In these orebodies, the chute samples give the information desired. In the sulfide orebodies, however, channeling is essential to determine the boundaries of "pay ore." Most of these orebodies are channeled before stoping commences and as necessity arises during the extraction period.

DIVISION I SOUTHWEST

Copper Ore Est. Jan. 1922

Orebody	Total Tons	Copper %	Copper Tons	Sulfur %	Sulfur Tons	Iron %	Iron Tons	Lime %	Lime Tons	Silica %	Silica Tons	Alumina %	Alumina Tons	Oz. Gold	Oz Silver	S/Cu
Total																

Fig. 1.

Assay maps show all channel samples and a comparison of chute and belt samples and mine and smelter samples made up for each 15-day period. The average difference between the chute and belt samples is about 0.4 per cent.; and between the shipment samples taken at the mine and smelter, about 0.2 per cent. on a copper content of about 7 per cent.

Ore-estimating Methods

At first, ore estimates were prepared by the engineering department and considered two classes of ore: ore in sight and probable ore. The first class was calculated by extending known faces of ore 10 ft. into the solid. The estimates of the second class of ore were based mainly on the judgment of the man making the estimate. Grades were given in round figures and again represented the opinion of the estimator. In 1913, the ore estimation was transferred to the geological department, where the present reserves are calculated. Semiannual estimates are prepared on Jan. 1 and July 1, but estimates of various orebodies or areas are made when desired by the management.

For the calculation of ore reserves, the detail figuring is done in bound books 10 by 12 in.; the completed calculations are transferred to

SUMMARY (1)

Smelting Copper Ore (not including miscellaneous) Est. Jan 1922

Division	Total Tons Jan 1, 1922	Change Since Jan 1, 1921	Change Since July 1, 1921	Copper %	Copper Tons	Sulfur %	Sulfur Tons	Iron %	Iron Tons	Lime %	Lime Tons	Silica %	Silica Tons	Alumina %	Alumina Tons	S/Cu
U.S.																
S.W.																
II																
VII																
W.T.D.																
Total																
Lean Sulfide Ore																
III																
V																
VI																
Total																
Grand Total																

FIG. 2.

cards, 5 by 8 in., one card being used for each orebody. Summary cards, Figs. 1 to 5, one for each division of the mine, are also

SUMMARY (2)

Lead-Silver Ore Est. Jan. 1922

DIVISION	HIGH GRADE Tons	HIGH GRADE Change since 7/1/21	GOOD GRADE Tons	GOOD GRADE Change since 7/1/21	LOW GRADE Tons	LOW GRADE Change since 7/1/21
Uncle Sam						
Southwest						
Czar						
Total						

Orebody	Total Tons	Copper %	Copper Tons	Sulfur %	Sulfur Tons	Iron %	Iron Tons	Lime %	Lime Tons	Silica %	Silica Tons	Alumina %	Alumina Tons	Oz. Gold	Oz. Silver	S/Cu
Gardner Silica																

Miscellaneous Smelting Copper Ore

Place	Tons	Change Since July 1921	Change Since Jan 1921	Copper %	Copper Tons
Tuscarora End Line (Oxide)					
Tuscarora (Enr. End Line (Sulp)					
Tuscarora (Prim. End Line (Sulp)					
Total					

Boras and Night Hawk Leases

Place	Tons	Change since 7/1/21	Copper %	Copper Tons	% Pb	Oz. Gold	Oz. Silver
Boras							
Night Hawk							
Total							

FIG. 3.

made up, as well as general summary cards for the entire mine. The latter show not only the tonnage remaining in the mine but also the change since the previous estimate and previous year, the footage of

development work, tonnage mined, total tonnage developed, and tons mined and developed per foot of development work done.

Details of Estimating Practice

The general principle underlying the calculation of ore reserves is a critical study of the individual orebody under consideration, its history, present condition, and geological relationship. The method used for

SUMMARY (4)

Limestone Mine Ore Reserves and Footage of Development Work - Year 1921 Est Jan. 1922

Division	Reserves		Change since 1/1/21		Tons Mined		Total Tons Devl'pd		Footage		Tons Devl'pd/ft		Tons Mined/ft	
	Copper	Lead	Copper	Lead	Copper	Lead	Copper	Lead	Copper	Lead	Copper	Lead	Copper	Lead
U.S.														
S.W.														
II														
VII														
W.T.D.														
Total														
Misc.														
Gardner Silica														
Total														
Lean Sulfide														
Grand Total														

Fig. 4.

SUMMARY (6)

Sacramento Hill Ore Est. Jan. 1922

Orebody		Smelting Ore					Milling Ore					Low Grade Ore					Leaching Ore				
		Reserve	% Cu	Change since July 1921	Tons mined	Tons devl'pd	Reserve	% Cu	Change since July 1921	Tons mined	Tons devl'pd	Reserve	% Cu	Change since July 1921	Tons mined	Tons devl'pd	Reserve	% Cu	Change since July 1921	Tons mined	Tons devl'pd
Inside Pits	West																				
	East																				
	Total																				
Outside Pits	West																				
	East																				
	Total																				
Total	West																				
	East																				
	Total																				

Fig. 5.

estimating the reserves in the limestone orebodies is determined by intimate knowledge of the orebody. Sacramento Hill porphyry deposit is estimated separately and by a different method. The methods are:

1. From horizontal sections, where the total volume of orebody is calculated, the tonnage mined is deducted to obtain the reserve.

2. From horizontal sections where only the ore remaining is estimated.
3. From vertical sections where the total volume is calculated and tonnage mined deducted.
4. From drifts and raises that cut ore in which no other work has been done.
5. From the sill floor of a stope with no work done below it.
6. From stoping operations of other companies on the property line, but with no work done opposite it.
7. Where orebodies are following the same porphyry dike or fractures.

Factors

In the construction of sections for estimating, all available information from drifts, raises, winzes and intermediate work is used. Not only is account taken of sampling results but geologic features and their relationship to the ore are considered.

The first method is used where the orebody is fairly regular and where the tonnage mined can be easily obtained.

The second method forms an important part of the estimating practice as a large number of orebodies are calculated by it at present. In using this method, the area of ore remaining at any elevation is taken for a height corresponding to the floor of the stope at which it occurs, for example, sill floor 10 ft. and stope floor 8 ft.

In the third method, where possible, vertical sections are spaced at intervals, up to 50 ft., depending on the location of raises. Areas of ore are not connected up between levels unless raises have gone through practically on the section. To obtain the reserve, the tonnage mined is deducted from the total volume of the orebody. This method is the most desirable on account of irregularities in vertical extent of the orebodies and is used whenever the necessary information is available.

Where ore is encountered in drifts or raises, it is figured as extending for 10 ft. beyond all free faces, unless known to end within that distance. In the past this has meant a block 28 by 25 ft. for drifts, taking the length of ore in the drift for the third dimension, and 25 by 30 ft. for raises, for the distance the raise was in ore.

Where a sill floor of a stope occurs with no work below, the sill area is projected downward 10 feet.

Where ore is mined by the other companies up to the property line and no work has been done opposite it, the area on the line is projected 20 ft. into Copper Queen ground.

In some few places, orebodies that are following the same porphyry dike or fracture, are connected up for distances not exceeding 100 ft. Records show that this method was used much more in the past than at present.

The weight of the ore is taken from actual weights of broken ore, determined at intervals by the engineering department and applying an expansion factor of 1.6 to 1. By averaging the weights obtained since estimating by the geologic department started, 9 years ago, a fairly accurate figure has been secured, which is not changed according to individual weighing of ore. This weight factor varies from 0.057 to 0.110 and is applied to the estimated volume to reduce it to tonnage figures.

Ore grades are figured on past production of each orebody. For this purpose the original estimate made by the geologic department was given a grade based on an average of all chute samples taken. In later estimates, this figure has been corrected by averaging with it the grade mined in the 6 months preceding the estimate. By disregarding the tonnages represented in the two figures averaged, it is assumed that the ore remaining will be more nearly of the grade just previously mined than that of preceding periods. When using this method for ore reserves, the economic aspect of the operations of the previous period must be considered. In other words, it is necessary to consider whether especially rich portions of the orebody have been mined, or whether the grade obtained can be assumed to be more or less that of the entire deposit.

In one section of the mine, the grade of the orebody is calculated from channel samples alone. This is an extension of the disseminated ore of Sacramento Hill that will not be included in the steam-shovel operations. The tonnage in this orebody is calculated from vertical sections and the grade arrived at by taking a careful geometrical average of the samples on each section. Two sets of sections, at right angles to each other, are used and the results obtained from the two estimates averaged for the final figures. Few samples used with one set of sections appear in the estimate from the other set, but the results have been found to be remarkably close both as to tonnage and grade.

Conclusion as to Underground Sampling

To summarize the foregoing, it might be said that Copper Queen experience has shown that for orebodies of the type found in the Warren district, the following points apply:

1. Systematic sampling of oxide and carbonate orebodies is not essential.
2. Massive sulfide deposits must be carefully sampled to determine the boundary between ore and waste.
3. Careful and detailed sampling of disseminated orebodies is a necessity.
4. Vertical sections form the most accurate means of estimating.
5. All available geologic information must be used to obtain accurate figures on reserves.

6. Only about 3 years' supply of ore, in limestone replacement deposits, is developed ahead of stoping on account of the difficulty and expense of maintaining openings in them and loss of interest on money involved.

7. Ore estimates are of more value as showing the comparative condition of the mine than its probable life.

Disseminated Orebodies in Sacramento Hill

Churn Drilling

Preliminary to the systematic development of Sacramento Hill with churn drills, an accurate contour map of the region was made on a 50-ft. scale with a contour interval of 5 ft. This map was laid out in 100-ft. squares. Drilling was started as nearly on the corners of these squares as the nature of the ground permitted.

During the process of drilling, water was poured into the hole to form a sludge with the cuttings. Material samples were taken at intervals of 5 ft. and all of the cuttings from the 5 ft. went into the sample, whether the sludge was bailed out more than once or not, while the distance was being drilled. The sludge was discharged into a launder leading to the Jones splitter, which by making three cuts of the sludge removed one-eighth as a sample. This sample was then placed in the upper compartment of a steam drier and the water removed by evaporation; heat was furnished by the boiler of the drill. (This method proved more satisfactory than using fire, as the temperature was never so high as to change any of the constituents of the ore.) After drying, the sample was sent to the assay office where it was ground and quartered for determination.

Among the errors peculiar to this method of sampling are: The tendency of the hole to cave; the concentration of heavy minerals in the bottom of the hole; and the deflection of the hole from the vertical. The first of these was largely remedied by casing the hole when it was found necessary. The error arising from the concentration of heavy mineral in the bottom of the hole was negligible, because the cuttings were so thoroughly mixed by the drilling tools that the heavier particles had practically no time to settle out before the sludge was removed by the bailer. Also, when each sample was taken the hole was cleaned as thoroughly as possible. As nearly as has been determined, there has been little difficulty through holes deflecting from the vertical; several holes were checked where they penetrated underground workings and the deflection was very little, considering the depths of the holes drilled. As several of the drill holes pierced underground workings, it has been possible to check the accuracy of the churn-drill method of sampling by driving raises along the holes. The general results obtained from these

raises agree closely with those obtained from the drill holes, as shown by Table 1.

TABLE 1.—*Comparison of Copper Assays from Churn-drill Sampling in Hole D-22, and Channel* Sampling in 2-80-20 Raise 2-157 Drift and 2-57 Winze (Only Adjacent Samples Are Averaged)*

Elevation of Sample, Feet	Churn Drill Sample, Per Cent.	Channel Sample, Per Cent.	Point of Channel
5195–5200	1.64	1.7	2 ft. north from drill hole in 2-80-20 raise
5185–5190	3.66	4.0	2 ft. north from drill hole in 2-80-20 raise
5180–5185	3.72	2.5	2 ft. north from drill hole in 2-80-20 raise
5175–5180	4.18	3.4	2 ft. north from drill hole in 2-80-20 raise
5170–5175	2.38	5.6	2 ft. north from drill hole in 2-80-20 raise
5165–5170	5.78	4.5	2 ft. north from drill hole in 2-80-20 raise
5160–5165	4.25	3 7	2 ft. north from drill hole in 2-80-20 raise
5155–5160	4.12	4.7	2 ft. north from drill hole in 2-80-20 raise
5150–5155	2.82	2 8	2 ft. north from drill hole in 2-80-20 raise
5145–5150	3 30	2.9	2 ft. north from drill hole in 2-80-20 raise
5135–5140	6.62	6.6	In 2-157 drift
5120–5125	3.68	6.2	1 ft. north from drill hole in 2-57 winze
5115–5120	3.65	6.5	1 ft. north from drill hole in 2-57 winze
5110–5115	5.75	3.8	1 ft. north from drill hole in 2-57 winze
5105–5110	5.70	5.0	1 ft. north from drill hole in 2-57 winze
5100–5105	5.52	2.2	1 ft. north from drill hole in 2-57 winze
5095–5100	2.50	1.1	1 ft. north from drill hole in 2-57 winze
5090–5095	1.92	0.6	1 ft. north from drill hole in 2-57 winze
Average.....	3.95	3.77	

* The channel samples taken for comparison with the churn-drill ones, are those nearest the hole.

Estimating Ore Reserves

From the information furnished by the assay office and the sampler at the drill, a log of each hole was made; this showed the position and value of each sample in depth. The ore in each hole was grouped into classes according to grades, as follows: Samples that assayed better than 3½ per cent. copper, "S" ore; from 1¾ to 3½ per cent., "E" ore; from 1.3 to 1.75 per cent. "C" ore; from 1 to 1.3 per cent. "F" ore; from 0.8 to 1.0 per cent., "L" ore; from 0.5 to .08 per cent., "P" ore. With these logs as a guide, cross-sections of the orebody were made both ways through the orebody at intervals of 100 ft. along the sides of the squares before mentioned. If a hole did not come exactly on the section, but was several feet away, the information contained in the hole was projected to the section and recorded there, the distance and direction of the hole being noted. These cross-section sheets were used to show the

general form of the orebody, and the relation of the different classes of ore to each other and to waste intrusion. They were not considered accurate enough for use in making the final estimates of ore tonnages; these estimates were made by the triangular-prism method, as follows.

Triangular Prism Method

Using the cross-section sheets, together with the logs of the holes as a guide, a triangular plat on a 50-ft. scale was made, the apex of each triangle being a churn-drill hole. Considerable care was necessary when making these divisions, so that each triangular prism would be equally valued. This divided the whole orebody into a series of triangular prisms, the edge of each prism being marked by a churn-drill hole. From the log of each drill hole, the ore reserves were calculated; a page from the book used in these calculations is here shown.

LOG OF HOLE

Hole No. 16

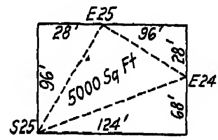
Elevation of collar, 5458 ft., depth, 555 ft.
Depth to be stripped, 400 ft.

DEPTH, FEET	CLASS	GRADE, PER CENT.	FEET		ASSAY FT.
				—S—	
160	Ox				
50	S	5.99	50	5.99	299.50
15	E	2.80	30	5.78	173.40
30	S	5.78			
40	E	2 32	80		472.90
10	G	1.43		—E—	
25	L	0.86	15	2.80	42.00
15	F	1.16	40	2.32	92.80
15	E	2.13	15	2.13	31.95
5	F	1.21			
15	P	0.43	70		166.75
15	G	1.42		—G—	
5	L	0.80	10	1.43	14 30
45	P		15	1.42	21.30
5	L	0.80			
85	P		25		35.60
5	F	1.00		—F—	
15	P		15	1.16	17.40
			5	1.21	6.05
555					
			20		23.45
				—L—	
			25	0.86	21.50
			5	0.80	4.00
			30		25.50
				—P—	
			15	0.43	6.45
				Strip	
			160		

The number of feet of each class of ore is multiplied by its assay value, which gives the feet per cent. of each class of ore. The total footage of each class of ore divided by its feet per cent. gives the average value of that class of ore in the hole. Three of these holes form the edges of a triangular prism. Each triangular prism is given a number and its volume and grade calculated separately. The following figures show the calculation.

	Lat. Dep.
E 25	9359–7002
E 24	9331–7098
S 25	9263–6974

Area 5000



	—S—		—E—		—G—		—F—	
	Feet	Feet, Per Cent.	Feet	Feet, Per Cent.	Feet	Feet, Per Cent.	Feet	Feet, Per Cent.
	15	138.90	35	87.60	15	23.90	30	34.80
							20	21.55
	5	30.50	10	20.85	15	22.85	40	43.35
	20	169.40	45	108.45	30	46.75	90	99.70
	6.67	56.47	15.00	36.15	10.00	15.58	30.00	33.23
Vol.		33,350.00		75,000.00		50,000.00		150,000.00
Vol. assay		282,350.00		180,750.00		77,900.00		166,150.00

	—L—		—E—		—P—
	Feet	Feet, Per Cent.	Feet	Feet, Per Cent.	Strip
	30	26.20	100	58.50	55
	70	56.35	20	13.00	90
	40	35.70	150	90.45	20
	140	118.35	270	161.95	165
	46.67	39.45	90.00	53.98	55.00
Vol.		233,350.00		450,000.00	275,000.00
Vol. assay		197,250.00		269,900.00	

In this calculation, the number of the prism, the holes forming its edges, and the area of the prism are shown. The first three lines of figures show the feet and feet per cent. of each class of ore in each hole. The average of these multiplied by the cross-sectional area of the prism gives the volume and volume assay of each class of ore. Dividing the volume assay by the volume gives the average assay value of each class of ore. The results from each triangular prism are tabulated and summed up as in Table 2, which gives the total volume of each class of ore together with its assay value.

TABLE 2.—*Total Inside* Δ's

Δ	S		E		G		F		L		P		Strip
	Feet	Assay Feet	Feet	Assay Feet	Feet	Assay Feet	Feet	Assay Feet	Feet	Assay Feet	Feet	Assay Feet	Feet
143	33,350	283,350	75,000	180,750	50,000	77,900	150,000	166,150	233,350	197,250	450,000	269,900	275,000
144	17,376	91,158	78,270	199,171	78,270	118,814	191,344	209,137	243,524	207,833	408,726	240,028	321,794
145	37,292	160,350	83,865	208,992	93,202	141,173	158,393	177,683	251,595	212,849	260,932	153,641	437,943
146													
[147													

In converting the volume into tons it was found, after determining the specific gravity of the ore, that for "S," "E," "G," and "F" classes of ore, 12 cu. ft. equaled 1 ton, and for "L," "P" and "waste," 13 cu. ft. equaled 1 ton. These factors were used in determining the tonnage of each class of ore and also the total tonnage.

Summarizing the above, holes were drilled through the ore body at the intersections of 100-ft. squares and these holes sampled at 5-ft. intervals. Logs of the holes were made showing ore by grades and classes. The orebody was divided into a series of triangular prisms, the edges of each prism being churn-drill holes. The average value of each prism was calculated separately and the results summed up for total ore in the orebody.

Methods of Sampling and Estimating Iron Deposits

One of the papers under this head covers the sampling and estimating of iron ores of the Lake Superior region as a whole while the other four papers cover the Gogebic, Marquette, and Menominee Ranges of Michigan and the Cuyuma Range of Minnesota.

Sampling and Estimating Lake Superior Iron Ores

By J. F. Wolff,* Duluth, Minn., E. L. Derby,† Ishpeming, Mich., and W. A. Cole,‡ Minneapolis, Minn.

(New York Meeting, February, 1922)

Exploration of Lake Superior iron ores is done principally by drilling. The soft iron ores are churn drilled and the harder ores are diamond drilled. The churn drill used is the Mesabi type and its operation should be described as "water jetting." The drill rods are washed down by the combination action of the water jetting from the bit at the lower end of the rods and the churning motion given the line of rods. The water, which is forced down through the rods, comes up between the wash rods and the casing, bringing the choppings to the surface. It is caught in three or more barrels, each of which has a stoppered hole 8–10 in. from the bottom. These barrels are successively filled, allowed to settle, and drained to the hole, until the desired depth is drilled. The sediment from each barrel is transferred to a settling tub in which it is allowed to settle; the water is then drawn off and the wet sample is collected and dried. The number of barrels used depends on the character of the ore. For example, when drilling sandy ores, all the wash water is caught and settled for the proper length of time without refilling any of the barrels. Thus, the sediment held below the stoppered hole in any one barrel is not disturbed.

The casing is kept as near the bottom of the hole as practicable to prevent the caving of the hole from jeopardizing the sample. While the casing is being driven down and the hole is being washed clean after casing, the return water is not run into the sample barrels.

In exploratory work in the harder formations, where the diamond drill is used, the sample comprises both core and cuttings. The care given in handling and preserving the core depends on the importance that may be attached to certain characteristic strata in different formations. The following instructions for the care of core samples are issued by the Oglebay Norton Co. for diamond drill work done for them on the Gogebic Range of Michigan:

* Mining Engineer, Oliver Iron Mining Co.
† Geologist, Cleveland-Cliffs Iron Co.
‡ Chief Engineer, E. J. Longyear Co.

The driller should be careful when removing the core from the core barrel to keep the core in exact order. Not only should the pieces of core be kept in exact relation to one another, but care should be used not to turn them end for end. Core should be placed in the core box from left to right in each row. Each drill should be furnished with small, hinged cover core boxes, and each drill should be well supplied with empty boxes. Core remaining at the drill when the drillers are not working should be locked up. The practice of picking sample pieces of core and carrying them away from the drill to be examined in better light, or shown to some one else, is unnecessary and prohibited. The reason for this is that these pieces are seldom returned to the core boxes and valuable information or "markers" may be lost. If the inspector suspects that a critical point is reached in the drilling and he cannot determine at the drill, he should remove the entire core box and see that the drill is immediately furnished with an empty core box.

It is customary to split the core longitudinally so that one-half may be ground to a pulp for analysis and the other half may be kept for future inspection. The core should be filed in permanent core boxes, properly labelled, if the core is to be kept in exact order as taken from the core barrel; otherwise, it is often filed in tin boxes in the same manner as sludge samples. To obtain the average analysis, the core and cutting samples obtained in the same run are analyzed separately and the analyses are combined in the proper ratio.

Collection of Cuttings

The cuttings are, as a rule, collected in a sludge or "floor box." Various types of boxes are used, but the one that may be recommended has the following dimensions: 6 ft. 1 in. long, 18 in. wide, and 8 in. high. It is divided into three compartments by baffle plates. It is made of 1-in. spruce, with a double bottom, and is lined with No. 26 galvanized iron. Iron tie-rods strengthen the construction of the box. Both ends of the box are closed but the lower end has two stoppered outlets.

The sludge box rests on the drilling platform, or floor, with the casing of the drill hole projecting through the bottom of the box near one end. Packing is placed around the casing to make a water-tight connection. Sufficient grade is given to the box to direct the overflow to the lower end, the height of water that may be retained in the box being approximately 6 in. The return water from the hole carrying the cuttings passes through the floor box and a partial settling of the suspended material takes place. After the footage representing the interval to be sampled, usually 5 ft., is drilled, the hole is pumped clean, the baffle plates are removed, the water is drained off through the outlets in the lower end of the box, and the sample is collected, dried, sacked, and labelled.

The floor box was designed after a large number of tests were made by the writer. A similar floor box with a 4½-in. head, rather than 6-in., gave an efficiency, based on analyses, of 98.22 per cent. and an efficiency by weight of 71.50 per cent.; that is, although less than three-fourths of

the total weight of material washed from the hole was caught by the box, the iron analysis of the floor-box sample of cuttings, for the average grade of ores, would probably be not over 1 per cent. too high or too low in comparison with the true iron content of all of the material washed from the hole.

Sampling Underground Development Work

Drifts and crosscuts in iron-ore deposits are usually sampled in 20 or 25-ft. sections. The number of places in which material is taken for a sample of each section depends on the character of the ore formation. The samples are generally taken as follows:

1. Each section is sampled at four stations; stations 1 and 3 are on one side of the drift and stations 2 and 4 are directly opposite. The relative position of the stations is maintained in each section. The sample representing each section is obtained by cutting at each station a groove 3 in. wide and 2 in. deep across the entire face of the ore at right angles to the formation.

2. Grooves are cut between each set of timbers, 5-ft. intervals, on opposite sides of the drift.

The material is caught in a pan or box, as the grooving is done, and a proper amount is taken, after mixing for the sample.

Raises, winzes, etc. are sampled in 5-ft. sections where the ore is uniform. If rock or lean ore occurs with the ore in the same 5-ft. interval, representative samples are taken of each material. If the ore lies in nearly horizontal strata, opposite or adjacent walls are sampled, otherwise the four walls are sampled. The walls to be sampled are first cleaned and a vertical groove, 3 in. wide and 2 in. deep, is cut down the middle of the wall from top to bottom of each 5-ft. section. The material removed is caught in a pan or box and the samples from the two or four cuts, as the case may be, are combined.

Iron Ore Estimates

Altogether when estimating ore it is usual to follow general empirical rules, it must be kept in mind that each deposit offers special problems for the engineer to solve. In all estimates the main factor influencing the results is the personal element, good judgment.

As ore estimating is a problem in structural geology combined with engineering accuracy, we will first briefly review the essential structural features of iron ore deposits and later discuss the rules formulated for purposes of conservatism. In the Lake Superior region, iron ores are the result of subsurface alterations of richer layers of the iron-bearing rocks and are localized at places where these alterations have been most effective. The structural features that are factors in such secondary concentration of iron ores, and therefore determine the shapes of the

ore deposits, are joints, faults, folds, intersection by igneous rocks, impervious sedimentary layers or igneous masses within or below the iron-bearing formations, and area of exposure.

In the Mesabi district of Minnesota, the localization of the ores by fractures in the iron-bearing formations appears to be the outstanding feature. Other structural features of prime importance, on the Mesabi, are the area of exposure and the occurrence of the iron-formation in characteristic horizons, as correlated by the engineers of the Oliver Iron Mining Co.

In other Lake Superior iron-ore districts, the structural feature that is more conspicuous in determining the location of ores within certain portions of the iron-formation is the impervious basement, usually having the configuration of a pitching trough. Impervious basements for the orebodies may be formed (1) by the intersection of the foot-wall quartzite with an igneous dike, as in the Gogebic district; (2) by irregular intrusive masses of basic igneous rock, as in the Marquette district; (3) by dolomite formation, as in the Menominee district; (4) by slate, as at the lower horizons of the Negaunee formation in the Marquett, Crystal Falls, Iron River, and Florence districts, and at the upper horizon of the Vulcan formation in the Menominee district; (5) by slate layers within the iron-bearing formation, locally developed in the Gogebic, Cuyuna, and Mesabi districts; and (6) by granite, as in the Gwinn district and very locally in the Mesabi district.[1]

Methods of Estimating

When estimating iron ore tonnages, whether from drill-hole explorations or from records of mine workings, two methods are employed, the average depth and area method, and the cross-section method, with application of the "end area" formula. If the orebody is one of the deep trough type, common to the Mesabi Range, or in a steeply dipping formation, common to the other Lake Superior iron ranges, the cross-section method is preferable. If the orebody is one of the flat-layer type, with a comparatively large horizontal extent and relatively uniform thickness, as are many Mesabi orebodies, the average depth and area method may be used. Probably most of the original total ore estimates of Mesabi orebodies have been made according to this method, particularly if only drill exploratory data were available.

The initial exploration of the iron orebodies of the Mesabi Range is done with churn and diamond drills. Different systems of locating the holes are used by different companies depending somewhat on the part of the Range to be explored and other local conditions. The most

[1] C. R. Van Hise and C. K. Leith: Geology of the Lake Superior Region. U. S. Geol. Survey, Monograph 52 (1911).

common system is to locate holes at intersections of north-south and east-west coördinates, spaced either 200 or 300 ft. apart. Holes are drilled between these if necessary to determine more accurately the outline and structure of the orebody. Experience has proved that the 200-ft. spacing is not too close for most parts of the Range.

Average Depth and Area Method

While the average depth and area method may be used in estimating the tonnage of ore, the method is not recommended except for flat orebodies having a fairly uniform thickness of ore. On a map showing locations of all drill holes an outline is drawn at an arbitrary distance, with 100 ft. as the maximum, outside of the holes that delimit the orebody; this outline represents the area of the orebody. Another plan is to place such boundary a distance outside of the limiting holes equal to the depth of ore in the holes, and to connect all points so established with straight lines. Another plan is to assume ore to extend 100 ft. beyond the limiting holes if the depth of ore in the hole is 50 ft. or more, to extend 50 ft. beyond if the depth is 25 to 50 ft., and to extend 25 ft. beyond if the depth is less than 25 ft. All three plans are purely empirical to establish uniformity of methods in the estimates of any one company and are probably conservative on the average. However determined, the ore area is obtained by a planimeter from the map and reduced to square feet. This area, multiplied by the average depth, in feet, of ore in the area disclosed by exploration work, gives the volume of ore in cubic feet, which is converted to tons by dividing by a suitable factor (discussed later). If the drill holes were spaced irregularly, as in much of the early drilling on the Range, it is advisable to group certain holes in computing average depths.

Cross-section Method

The cross-section method is much more accurate than the foregoing. Four main and several minor structural subdivisions can be recognized in Mesabi Range orebodies. In drill explorations, all drill holes, both in rock and ore, should be classified closely into these known structural subdivisions and such classifications correlated on cross-sections. This work can be done only by engineers expert in such classification. Such structural layers will give indications in different places on the cross-sections of proximity of the rock walls that enclose the orebodies. Holes can be drilled between the holes spaced 200 or 300 ft. apart where required to determine more accurately the locations of rock walls enclosing the orebody. Especially in the case of the trough-type orebodies, the limiting rock walls continue for long distances in fairly straight lines and can be traced between drill holes with considerable accuracy by proper loca-

tion of intermediate drill holes, following close structural study of the original holes spaced on intersections of coördinates.

The limits of the orebody can be transferred from the drill-hole map to the vertical cross-sections, after they have been established by this study and then estimates of volume made from the cross-sections. The volumes are converted to tons by use of a suitable factor. The interval between cross-sections is usually 100 ft. unless the irregular character of the orebody makes it advisable to have them closer. Such estimates should be made by separate structural layers, because of the regular shape the drill holes often show the orebody (especially its bottom) to have. The average analysis or quality of the ore is computed on each cross-section for each structural layer separately by the foot-unit method. Then the average analyses of similar class ores represented on each cross-section are combined in proportion to their tonnages.

A variation of the cross-section method, with horizontal instead of vertical sections, is often used in estimating ore in a developed mine, particularly if mining is well advanced and the orebody is considerably cut up by mined areas and pillars. The horizontal sections used are the plan maps of the mine levels and sublevels.

Empirical Rules

The foregoing methods of estimating iron ore are used in the various fields with certain variations, which may be indicated by the following empirical rules adopted to guide the engineer in his interpretation of geological data:

1. In the Mesabi district, the edge of the orebody is placed at a distance that is a function of the depth of ore in the drill hole, or at an arbitrary distance with 100 ft. as the maximum. Data should be available at 300 ft. intervals to establish the continuity of the orebody. However, if the structure of the orebody is known, the continuity of the ore in its horizontal extent may be considered to be established for a much greater distance than 300 ft. between proved points in the ore deposit along the trend of the major axis of the orebody; that is, trend of system of fractures that directed course of concentration. At right angles to this trend, the 300-ft. interval is often too great.

2. In the Cuyuna district, ore is usually estimated from the surface material down the dip to a horizontal plane at the greatest depth of ore encountered on each cross-section, although some operators assume the ore to continue 50 ft. below the deepest ore cut by drill holes. At right angles to the bedding, very little leeway (less than 25 ft.) is allowed in estimating the ore in a stratum not actually cut by a drill hole or development workings. Ore is considered to be proved between cross-sections at 300 to 400-ft. intervals along the strike.

3. In the Marquette district, estimates are made by the vertical cross-section method or level-plan method.

If the structure controlling the localization of the ore is determined by drilling or development work, the ore at the ends of the cross-sections are limited by it. If the structure is not indicated, the ore is limited on the ends of each cross-section by a triangle, the altitude of which is one third the thickness of the ore at its proved edge.

The ore is assumed to continue along the strike, or line at right angles to the plane of the cross-section, beyond the end sections for a distance of 50 ft. with the same areas as it had on the end sections.

When using the plan method, the ore is limited above and below the extreme horizontal sections by apexing, pinching out, or extending with a full area a variable distance, depending on the size of that area and upon local conditions. Usually this distance is from 25 to 50 ft.

4. In the Vermilion district, considerable drilling has been done during the past forty years but the amount is relatively small compared with the drilling done in other Lake Superior iron districts. No large iron ore deposit has been discovered by initial drilling and the Vermilion Range is known as the most difficult and expensive area in which to drill in the Lake Superior district; consequently no practice for estimating ore discovered by drill holes has been established for the Range.

The iron formation or jasper is found in pitching synclines and anticlines, or in steeply dipping lenses, having greenstone foot wall and in many places apparently overlaid by greenstone also. Both jasper and greenstone are intruded most irregularly and intricately by basic igneous rocks, chiefly porphyries. Beside the two large orebodies at Soudan and Ely, only four or five small and (to date) unimportant orebodies have been discovered on the Vermilion. During the past year, considerable geolgic knowledge has been obtained, particularly in recognizing great quantities of intrusives in both Ely and Soudan mine areas and in interpreting accurately the structure of these orebodies. This additional information is of much importance in estimating ore tonnages.

Ore estimates are made from mine workings. Such ore as is practically blocked out underground is estimated as developed ore, either by the cross-section method or by averaging the ore areas of the levels and multiplying by the distance between. Probable ore is estimated also below bottom levels and as extensions laterally from the main body of developed ore where geologic conditions indicate the probable presence of additional ore. No fixed practice is followed in estimating such probable ore, except that 100 ft. vertically below a bottom level is the maximum depth to which ore is assumed to extend. This limit is warranted because of the presence of large quantities of igneous intrusive rocks in both Ely and Soudan orebodies and the uncertainty as to where the orebodies may be cut off entirely in depth by these intrusives.

The Soudan ore is very hard, dense, and high grade and 10 cu. ft. per long ton is used in estimating it. Some of the Ely ore along the foot wall is hard and dense, but much of the orebody is relatively soft; some is even granular and fine. When estimating 12 cu. ft. per long ton is used. The Ely orebody lies in a syncline one limb of which extends eastward between greenstone foot and hanging walls. The orebody in the synclinal area occurs at the base of the jasper on the greenstone foot wall. At one place, practically the entire width of the syncline is ore. The bottom part of the eastward extension of this jasper syncline is ore, having greenstone on either side of it. Ore is developed in the Ely trough to a vertical depth of 1460 feet.

The Soudan mine is in a combined anticlinal and synclinal area cut by many intrusives so as to form pitching troughs in the jasper. The ore has been localized by leaching and concentration of the jasper in these troughs, one of which has been followed to a vertical depth of 1436 ft. and the other to 1586 feet.

Classification of Ores

In drill explorations and mining work, most companies analyze ore samples for iron, phosphorus, silica, and manganese; some analyze for alumina also. In all estimates, whether made from drill-hole explorations or from mine workings, classifications of ore are made for commercial purposes, based on chemical and physical character of ore and estimates are made of the tonnage of each class of ore. The most common classifications of Lake Superior iron ores are the following:

Standard ores, most of which are of merchantable grade and can be shipped direct to docks.

Washable ores, occurring principally on Mesabi and Cuyuna Ranges.

Lean, non-washable ores.

Manganiferous iron ores.

All ores analyzing above 40 or 50 per cent. dry iron are classified as Standard ores; some companies use one limit, some the other. Standard ores are subdivided into bessemer and non-bessemer. A bessemer ore is one that can be used, with suitable fluxes, to make a bessemer pig iron. According to the current furnace practice on Mesabi Range ores, those ores are classified as bessemer in ore reserve estimates, that contain 0.00075 to 0.00085 per cent. or less of phosphorus for each per cent. of iron. The particular ratio to be used depends on the phosphorus content of the limestone and coke that furnaces use. Actual bessemer ore shipments are graded closer to the higher ratio than to the lower. All Standard ores not bessemer are classified as non-bessemer.

Washable ores are those in which fine free sand is interbedded with lenses and layers of clean ore. Washable ore classifications are made

from an examination of samples, the texture and structure governing more than the iron analysis.

Lean, non-washable ores are all churn-drilled material of non-washable character analyzing less than 49 or 50 per cent. dry iron and over 35 or 40 per cent. Different companies use different limits. It is the general practice to classify all the diamond-drilled material analyzing less than 49 per cent. or 50 per cent. dry iron, as taconite, or jasper, the local names for the iron formation rock.

Manganiferous iron ores have been shipped from the Lake Superior district for some years and in large quantities, particularly from the Cuyuna Range, since 1915. The manganese content of such ores has been from 3.50 to 30 per cent. and the iron content from 35 per cent. up, the minimum combined metals of such ores shipped analyzing about 53 per cent. dried at 212° F.

In addition to these classifications, ore from producing or prospective producing mines are classified into grades having definite trade names and guarantees under which they are shipped and sold.

Computation of Grades

The quality of ore of each class or grade in an orebody is computed by the foot units method; that is, each assay is multiplied by the footage it represents and the summation of the like units is divided by the total footage of samples. In drill explorations, samples are taken usually of each 5 ft. of material drilled.

This method of computing the average quality of ore can be applied to estimates made from records of mine workings, using drift, raise, winze and shaft records instead of drill-hole records, taking care to use in the proper portion the different records. For example, on the Mesabi on account of its comparative flatness a 25-ft. drift sample may be given equal weight in the calculations to a 5-ft. raise or drill-hole sample.

Cubic Feet per Ton Factor

The density of the ore varies considerably in different layers of any orebody and even in different parts of the same layer, depending on iron content, porosity, and hydration. The relation between iron content and density is fairly definite, but many wide variations from a fixed relation occurs. Wherever possible, density tests should be made in many different parts and different structural layers of an orebody by weighing the amount of ore excavated very carefully from a given volume. A steel frame 2 or 3 ft. cube can be used very conveniently for this purpose, driving it down into the ore and excavating the material enclosed as it is driven.

More than 20 years ago, the results of many such tests on Mesabi Range ores were tabulated and platted with relation to the iron content of

the ore and an average curve drawn showing such relation. This curve still is in wide use by mining companies and Minnesota State Tax Commission, especially for determining density of ore in bodies explored by drilling but not opened up.

The factor of cubic feet per long ton of ore in place in an orebody for Mesabi ore varies from 10.5 to 18 cu. ft., the average being approximately 13.5 for the direct shipping ore. The washable ore of the western Mesabi averages about 14 cu. ft. per long ton. An average of 20 cu. ft. may be used as the volume of crude ore which will make one ton of concentrates. This figure is based on the fact that in the Coleraine district of the western Mesabi, an average of 20.6 cu. ft. has been excavated per ton of ore shipped, including both concentrates and direct shipping ore. This figure includes all lean ore taken from open pits and stock-piled and is representative of several million tons of ore mined.

Where the ore formation is adaptable, the factor of cubic feet of ore per long ton can be determined, by means of moisture of saturation method as described by W. J. Mead.[2] This method is not generally used in the iron-ore districts, principally because the ease with which it can be applied is not understood. According to Mead,

> The cubic contents of an ore is a direct function of (*a*) true specific gravity of the material—that is, the specific gravity unaffected by porosity or moisture; (*b*) porosity of the material, in terms of percentage of volume occupied by pore-space or voids; (*c*) percentage of moisture in the material—that is, the percentage loss in weight on drying at 110° C.

By saturating the ore in place, a means is found of calculating the porosity of the ore. The specific gravity may be computed from the analyses and the moisture in the ore taken from the average of shipments or determined from a sample of the ore taken before saturation.

The samples are obtained by cutting a series of niches in the bank or along the sides of a drift. The niche is cut so that an undisturbed projection of the ore is left. Water is then poured into the niche and the knob or projection of undisturbed ore becomes saturated after sufficient time has elapsed with the niche filled with water. Samples are then taken of the ore *saturated in place*. Composite samples from a series of these niches are usually taken. These samples are analyzed, in case of iron ore, for iron, silica, alumina, moisture, and loss by ignition. In the following example the alumina content was so low that it was considered negligible, for purposes of calculation. The analyses were as follows: 59.14 per cent. iron, 3.97 per cent. SiO_2, 8.03 per cent. loss by ignition, 14.9 per cent. moisture after saturation, 11.0 per cent. moisture in natural state.

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[2] *Op. cit.* 480.

The mineral density or true specific gravity of the material is obtained by actual determinations by gravity methods, or is calculated by properly combining the specific gravity of constituent minerals. By the latter method, it is necessary to determine the mineral composition of the ore from its chemical analysis. In this case, the loss by ignition (8.03 per cent.) is considered to represent the water of crystallization of limonite, which is known to be one of the main constituents of the ore. Pure limonite contains 14.5 per cent. water, hence the ore is in part $\frac{8.03}{14.5}$ or 55.4 per cent. limonite and the mineral constituents of the ore, for our purpose, are taken to be: 55.4 per cent. limonite, 4.0 per cent. silica, 40.6 per cent. hematite.

The specific gravities of the principal minerals of iron ore are as follows: Hematite, 4.5 for earthy ores, 5.1 for crystalline and hard ores; limonite, 03.0; kaolin, 2.62; quartz, 2.65.

As specific gravity is a function of volume as well as weight, the mineral specific gravity is obtained as follows:

55.4 per cent. by weight has a volume equal to $\frac{1}{3.6}$ of an equal weight of water.

4.0 per cent. by weight. has a volume equal to $\frac{1}{2.65}$ of an equal weight of water.

40.6 per cent. by weight has a volume equal to $\frac{1}{4.5}$ of an equal weight of water.

Reducing to unit volumes:

Unit Volumes

$$\frac{55.4}{3.6} = 15.4$$
$$\frac{4}{2.65} = 1.5$$
$$\frac{40.6}{4.5} = 9.0$$
$$25.9$$

$\frac{100}{25.9}$ = 3.86 mineral specific gravity.

From analyses; 14.9 per cent. = moisture of saturation; *i. e.*, amount of water by weight necessary to fill all voids. Hence, 14.9 per cent. water (specific gravity 1) occupies 14.9 unit volumes; 85.1 per cent. with specific gravity 3.86 occupies $\frac{85.1}{3.86}$ = 22.0 unit volumes.

$22.0 + 14.9 = 36.9$ unit volumes.

$\frac{14.9}{36.9}$ = 40 per cent. of porosity.

By using the diagram in Monograph 52, an ore with mineral specific gravity of 3.86, 40 per cent. porosity, and 11.0 per cent. moisture, is found to occupy in place 13.7 cu. ft. to the ton (2240 lb.).

Mining Loss

In estimates made from exploratory data, no deductions are made for rock and mining loss as experience has shown that the minor extensions of the orebody, which cannot be determined in normal explorations, will offset such losses.

When estimating orebodies completely developed, this deduction varies with the mining methods employed and factors of a purely local nature.

Estimating on the Gogebic Range

By J. F. Wolff, Duluth, Minn.

(New York Meeting, February, 1922)

The iron formation of the Gogebic Range in northeastern Wisconsin and the northwestern part of the Upper Peninsula of Michigan, resembles that of the Mesabi Range in Minnesota very closely in lithologic character and the subdivisions and horizons that can be recognized. It contains numerous interbedded slaty layers and considerable intraformational conglomerate in certain horizons.

In contrast to the flat dip of the Mesabi formation, the Gogebic dips steeply to the northwest at an average angle of about 70°. It has a quartzite and siliceous slate foot wall and a hanging wall of conglomerate, graywacke and slate. The black and gray Tyler slate has been designated the hanging wall of the Gogebic iron formation in U. S. Geologicial Survey monographs, but between the true Tyler slate and the commercially important Gogebic iron formation, and apparently unconformably upon the latter, is a series 600 ft. or more thick of conglomerate, slate, graywacke, and cherty iron formation, the latter being of no commercial importance on the Gogebic Range as far as now known. This series is probably that described as the "Capps" on the eastern Gogebic by the Michigan Geological Survey.

The Gogebic iron formation proper can be subdivided into three main horizons, a lower cherty and an upper cherty with a banded or slaty horizon between them, corresponding to the three lower horizons of the Mesabi. Several minor subdivisions can be made of these major ones. Orebodies occur in all three horizons of the Gogebic series and in some places ore is developed across practically the entire series.

Dikes and Sills

The Gogebic iron formation has been intruded most intricately by basic dikes and sills, most of the former pitching east but some pitching west. One very extensive sill is intruded near the base of the lower slaty or banded horizon, in the formation known locally as the Yale slates. One extensive bedding or strike fault is associated with this sill and a great many transverse faults occur at different places along the Range. The combination, or association, of steep dip, faults, dikes and sills has produced conditions very favorable for the development of orebodies through the leaching, oxidation, and consequent concentration effected

by circulating ground waters. The intersections of the dikes with the foot wall of the formation and with different slaty layers and basic sills in it have formed pitching troughs that directed the flow of underground waters and localized the formation of orebodies from the iron formation. The largest orebodies are found on the east-pitching dikes, but several large bodies have been developed on west-pitching dikes. The intersection of east- and west-pitching dikes is favorable for the development of orebodies; some orebodies have developed along the transverse faults that cut the iron formation. Geologic evidence indicates that water has followed down the pitching dikes from the outcrop and also has entered the iron formation where it is shattered by bedding and transverse faults.

The troughs formed by the intersection of foot wall or interbedded slaty horizons and the dikes, or by intersections of dikes themselves or dikes and sills, are V-shaped. In length they vary from a few hundred to several thousand feet; and in width and height, from several feet to a few hundred feet. Several orebodies may occur one above the other in successive horizons of the iron formation on the same dike. Different dikes intersect the iron formation at varying depths so that any one 40-acre tract may contain several orebodies, on different dikes at different vertical depths. Ore is known to exist at a depth of approximately 3000 ft. The top part of the cross-sectional area of any part of an orebody usually is quite irregular in outline. The greatest vertical height of the cross-section usually occurs along the foot wall of the orebody. The tonnage in individual orebodies varies from a hundred thousand to tens of millions.

Methods of Estimating

Methods of estimating used on the Gogebic Range depend on the exploration method by which the ore has been proved. If the ore to be estimated has been developed by underground workings, some engineers estimate the volume included between different levels by multiplying average areas of adjoining levels by the average distance between levels. The depth to which ore is assumed to continue down a dike below the bottom level depends much on the size, shape, and quality of the orebody, but it is rarely estimated more than 100 ft. below the bottom level.

If underground work is extensive enough to allow of preparation of cross-sections every 50 ft. along the strike, estimates are made by cross-sections rather than by areas of levels multiplied by average depth between them. Many of the largest mines on the Range are estimated by the cross-section method. Deductions are made for mine workings, whichever method is used. If the orebody is uniform and large and there is no diminution of area on the end cross-sections, or other limiting

conditions, it is common to assume the ore to continue 100 ft. beyond the end areas that have been developed. If the orebody is large and regular, the ore is assumed to extend not more than 100 ft. below the bottom level unless the underlying dike lies within such limit, in which case the orebody would be figured as extending downward from the bottom level to the dike. If the orebody lies in pockets rather than in a large regular body, it is rarely assumed to extend more than 35 ft., or two working sublevels, below the bottom developed level.

Results of Diamond Drilling

Except in the area of the Plymouth-Wakefield orebody, west of Sunday Lake, Gogebic Range orebodies have not been explored or discovered by diamond drilling. Drills have been used principally to locate the position of dikes, foot wall, etc., rather than to discover and block out orebodies. The shape, regularity, and continuity of Gogebic Range orebodies varies considerably in the different horizons of the iron formation; orebodies derived from the slaty horizons are the most irregular. Mine operators, therefore, do not rely on or make estimates from drill results, unless the drilling is so complete as to establish the outline and continuity of the orebody beyond question. The Wakefield and Plymouth orebodies are practically the only ones on this Range that have been thoroughly explored and blocked out by drilling before starting mine developments.

When explorations are by drilling only, such drilling must be complete enough to determine the location of the dike upon which the orebody occurs and the foot wall, as well as the approximate outline of the top of orebody on each cross-section of the drill holes, in order to estimate tonnage of ore developed with any degree of accuracy. Usually only one or a few 40-acre tracts are explored in one exploration program, and the drilling must be extensive enough along the strike to determine the continuity of the orebody across the property in question, or else to delimit the ends of the orebody within the property. The distance to which ore is assumed to continue along the strike beyond the outside ore-holes depends entirely on the associated geologic conditions and is a matter of judgment rather than one of fixed standards used by all mining companies. Many engineers in the district would not assume ore to extend more than 50 ft. beyond limiting drill holes in ore. All geologic information available from adjoining properties is used in estimating tonnage on any property, whether developed by underground workings or drilling only.

Conversion Factors

Factors of 10 cu. ft. per long ton, for ore averaging over 60 per cent. dry iron and 11 cu. ft. for ore below 57 per cent. dry iron are used when

reducing volumes to tons. A 10 per cent. reduction usually is allowed for mining loss and 10 per cent. for rock encountered in the ore.

Average analyses for specific tonnages are computed by the foot-unit method and tonnage by the ton-unit method.[1] Classifications of ores also are made as explained in connection with the Mesabi Range, although at local mines classifications are made to conform to the particular commercial grades produced from the mine.

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[1] See page 649.

Method of Sampling Ores on the Marquette Range

By R. W. Bowers

(New York Meeting, February, 1922)

In order to hoist the ore according to grades, stope samples are taken. Three equidistant grooves are picked across the breast at right angles to the dip of the ore, the part removed being caught in the hand and then put into a small bag. When there is broken ore in the breast, the lines are continued down the pile and a small amount taken at intervals of the length of the sampler's pick handle. The quantity of sample from each breast will be between 2 and 3 lb.

If there are two grades of ore, bessemer and non-bessemer, there are two sets of brass checks for each chute, each check stamped with the number of the chute; one set of checks has the word "bessemer" and the other "non-bessemer." Before any ore is removed from the chutes, it is the duty of one man to see that the proper set of checks is at each chute in the mine. As each motor car is loaded, a check is attached to it. When the train reaches the shaft, the sample man removes the check on the first car, notes the grade, and directs the motorman to spot it over the proper pocket. The check is then hung on a hook with the number corresponding to the chute from which the ore came. The sample man throws a short rope with a knot in the end over the car and at the point where the knot falls a standard scoopful is taken. This is placed in a large bag that hangs directly under the hook containing the checks for that particular chute. These chute samples represent a check on the grading of the various contracts and by counting the tags a check is obtained on the miner's tally of the ore going into each chute; the total of the checks represents the hoist.

During the stocking season, a standard scoopful is taken from each top tram car. This sample is taken in the same way as the chute samples at the plats underground.

Top-of-car Samples

These samples are taken by using a rope, preferably a clothesline, knotted each 18 in. of its length, and a hammer and a scoop. The standard scoop measures 3½ by 2¼ by 1¼ in. and holds about ½ lb. of ore. The rope, as applied to the usual railroad cars at the mine, is knotted into fifteen sections. It is placed diagonally over the car from

end to end and a scoopful sample is taken at each spacing; this amounts to about 8 lb. of sample to a car.

Dipper Sample

The dipper sample consists of a standard scoopful of ore from one or more places from each dipper of ore loaded by the steam shovel at the stock pile. Ordinarily this sample is taken at random, the sampler determining by the appearance of each dipper of ore, the proper proportions of lump and fines to be included in the sample. On difficult ores, the lump and fines of which differ in any particular, the sample may be taken by the use of a rope, knotted at one or more places, which, when thrown over the ore, determines the proportion of lump and fines taken as the sample.

Skip Sample

The skip sample is taken at the pocket, from each skip loaded into the railroad car, either by the use of a scoop or a dipper. When taken with a scoop, each skip is sampled from the railroad car, from one or more places; if a knotted rope is used depends on the nature of the ore. When taken with a dipper, the sample is caught at the lip of the pocket chute, from the running ore, one dipper full from each skip dumped.

Samples from all the mines are accumulated in units of from one to twenty cars, depending on from how small a unit of loading the analysis is required. Pocket samples are usually accumulated for all cars loaded for a shift or for a day; with the exception of bessemer ores, for which it is sometimes found necessary to sample and analyze each car as a unit. On stock-pile loading, the sampling is done in five, ten, fifteen, or twenty-car units, and occasionally on separate cars; loading for which sample is taken and analysis made consists of fifteen-car lots of non-bessemer ores, and of five or ten-car lots of bessemer ores. In general, the dipper sample is taken instead of the top-of-car sample, if the unit of loading consists of less than ten cars to the lot. By dividing the day's samples into several lots, rather than one large sample representing the entire day's loading, the samples are kept down to a size convenient for handling, and the effect of any error, in taking the samples or in their subsequent handling, is confined to a smaller amount of ore loaded.

Ore Estimation on the Menominee Range, Including Iron River, Crystal Falls, and Florence Districts

J. F. WOLFF, DULUTH, MINN.

(New York Meeting, February, 1922)

THE iron formation of the Iron River, Crystal Falls, and Florence districts probably is not of one geologic age only but of at least two ages. The ores, therefore, are of different characteristics. But in general the geologic conditions that occasioned the development of the orebodies have been essentially the same throughout the district and the types and structures of the orebodies are very similar in different parts of the area.

In general, the iron formation of this area has been folded into a series of close pitching synclines and anticlines; minor folds of many orders are included in the major folds. The pitching synclines have been particularly favorable places for the circulation of ground waters, which in such troughs have developed orebodies through the familiar process of leaching out the silica in the iron formation, oxidizing and concentrating ferrous iron compounds in the rocks.

The iron formation of this area is interbedded with and has interbedded in it certain slate or slaty layers that, in addition to the foot wall of the iron formation proper, act as impervious layers in the pitching troughs, to direct and control the flow of underground waters. In most places, the entire iron formation in a structural trough is not altered into iron ore; the orebody may be lens-shaped and exhibit no folding in its layers, outline, or shape, but almost without exception such bodies have been developed in folds and because of the influence of folds, and structurally they occupy only one limb of a fold. Many orebodies occupy the entire trough of a minor fold and the structure of the fold can be traced throughout the orebody.

Many definitely recognizable horizon markers in or associated with the iron formation occur in this district and serve as valuable guides to the engineer or geologist in exploring an orebody whether by drilling or underground mining development. Among such markers are the talc schist (an altered dolomite) and the Traders slate at the base of old Menominee iron formation proper, and the interbedded Brier slates and overlying Hanbury slates; and certain graphitic slates and graywackel in the Iron River and Crystal Falls area.

Orebodies in this area vary in width from a score to a few hundred feet, depending on structure and point of measurement in a trough; they vary in length from a few hundred feet to more than a half mile; somer are known to extend to a depth of more than 1500 ft. The tonnage in individual orebodies varies from a few thousand to several million tons.

Methods of Estimating Ore

As in other parts of the Lake Superior iron district, the methods used in estimating ore tonnages depend much on the way in which the ore has been proved. If the ore has been proved by diamond drilling, the tonnage is estimated from geologic cross-sections of the drill holes. If it is proved and blocked out by underground workings, the plan area and average depth method is used, as described for Mesabi and Gogebic Range orebodies. Average analyses of the ore are computed as described for those ranges, also.

From the description of the geologic conditions that obtain in this area, it is apparent that either drilling or mine development must be complete enough to allow the engineer or geologist to outline the structure and the limits of the orebody, in order to determine the total tonnage in the body with any degree of accuracy. No arbitrary limits are adopted in the district in assuming ore to extend beyond proved ore either in drill holes or drifts. Such assumptions, if necessary, are determined by the geologic conditions present. It is uncommon to assume ore to continue more than 100 ft. below a bottom level or beyond an outside drill hole. When estimating the high-grade ores 10 cu. ft. per long ton is used, 11 cu. ft. for the lower grade ores of the old Menominee Range proper, and 12 cu. ft. for Iron River, Crystal Falls and other ores. Reductions of 10 per cent. for rock in orebody and 10 per cent. for mining loss are customary.

Estimating in the Cuyuna Iron Ore District, Minnesota

By Carl Zapffe, Brainerd, Minn.

(New York Meeting, February, 1922)

Because no rock outcrops exist in the Cuyuna district, it is necessary first to make a magnetic survey with a dip needle, or possibly with a sun-dial compass, and determine the area or belt of magnetic variations and the maximum lines of magnetic attraction. A dip-needle survey is quicker to make, is sufficiently accurate, and will suffice. Regardless of how poorly defined the magnetic attractions may be, it is important that they be determined. Even if drilling has developed ore on an adjoining property, the survey should not be omitted.

Iron-bearing formation is indicated by the magnetic attractions, but whether or not ore exists must be determined by drilling. The maximum line of magnetic attraction offers the best place to begin drilling. If no magnetic attraction exists, nearby developments are the only guide; and if these also are lacking, drilling is governed by guess.

Test-pitting is not employed in exploratory work because the surface is too deep. Tunneling cannot be used because the topography is not suitable. Exploration shafts are a waste of time and expense, because from every standpoint drilling with churn and diamond drills is the most applicable method for exploratory work.

Light drilling equipment only is needed. Rarely have heavier drills been required, for seldom do holes exceed 400 ft. in depth, and about 100 ft. of this is in glacial drift which is easy to penetrate. The deepest hole is 1023 ft.

Because of the steep inclination of the strata and the narrowness of the orebodies, many angle holes have been drilled, mostly at 60° to 70° from the horizontal and against the dip of the formation. Angle holes are most advisable. If after initial drilling the beds are of shallow dip, vertical holes or holes of any angle can be used. The total cost of drilling a property is much greater if only vertical holes are used because more holes must be drilled to obtain the requisite information than when angle holes are used.

Many tests for variation in angularity of a finished hole have seldom shown more than 2°; therefore, such tests may with safety be abandoned.

Because the orebodies are generally about 100 ft. wide and cross a 40-acre tract at a slight angle, and as the dips are usually about 70°

southeast, a general plan of drilling is always applicable. In a typical case, the magnetic survey has shown the trend of the formations and probable place for drilling. A model method of exploration is to establish four or five cross-sections 300 to 400 ft. apart, each section containing two or three angle holes at 60° or 70° directed northwest and 50 to 100 ft. apart. After that work is concluded, the cross-section intervals should be bisected; the additional holes should be drilled at any angle or vertically and in any number that appear desirable to give the required data for each gap. Each property has its variations, but that plan will yield the best results in most cases. Final sections should not be more than 200 ft. apart, and preferably should be less. Whatever method may be used, holes should not be placed promiscuously; they should be placed along definite lines of cross-section, otherwise it is difficult to plat the results, and in estimating the tonnage the promiscuous drilling loses tremendously in value. Many properties have been discredited because lack of system failed to furnish the desired proof. Distances between holes in cross-section should not exceed 100 ft.; it is better to use a 50-ft. interval. Ore has often been missed by making a 100-ft. jump.

Estimating Tonnage

The drilling should be mapped in plan and in cross-section. The plan shows the position of the drill holes; the direction of the holes, if angular, and the horizontal projection of the hole; and the rock surface of the various formations. Plans at other elevations than the rock surface are not necessary. Any scale will do: 100 ft. to 1 in. or 200 ft. to 1 in. is suitable. A separate sheet is used for each cross-section; it shows the drill holes, the rock classification, various depth-figures, analyses of all samples taken, and dip of the formations. The scale should preferably be not less than 1 in. is 40 ft. In the absence of dip readings from cores, the body should be assumed to dip southeast if other drilling shows nothing to the contrary.

It is assumed that the drill holes have been placed in rows, or lines, that are at right angles to the strike of the orebody. The strike is usually nearly a straight line. A few orebodies occur on the nose of a fold, when the strike varies considerably. The variations in strike become apparent as drilling progresses; then one changes his spacing and direction of the lines of cross-section.

Cross-sections should not be more than 200 ft. apart, which means that ore in a drill hole is considered as extending 100 ft. to each side of the hole. If adjacent cross-sections show substantial quantities of ore, one seldom errs in making that assumption of distance; if a drill hole shows a small stringer of ore, and if adjacent cross-sections do not show substantial quantities of ore, such stringers or bands of ore are not considered to extend more than 50 ft. on each side of the drill hole.

Unless the drilling shows special characteristics, it is safe to consider every deposit of ore to be shaped like a long, narrow wedge with its base uppermost. Even if folding has developed a basin-shaped deposit and erosion has removed much of the structure, the deposit will exhibit greater length than width; but then the bottom of the deposit will be broad and even rather than knife-edged or fringed. These characteristics make it safe to consider the space between two cross-sections as nearly regular shaped prisms, or pyramids, or wedges the two end-areas of which can be measured with a scale or a planimeter. From the two end areas, the average- or mean-area, should be calculated and this area multiplied by the distance between the sections measured along the strike. The sum of all such segments in a tract gives the total number of cubic feet of ore.

For odd-shaped portions or isolated patches of ore, no rule can be laid down; the estimator must use his own judgment as to how much weight to give the ore shown. Small stringers should not be estimated, for they seldom pay to mine.

Should the cross-sections be over 200 ft. apart and if the distribution of the ore appears to be regular and the sections show only a little interbedded lean material, one is justified in allowing greater lengths between cross-sections. In many instances 300-ft. intervals proved acceptable for estimating because drilling was carried on in a regular manner. No doubt before these properties are mined the intervals will be bisected.

If deposits contain much interbedded lean material, a total of the square feet of all bands of ore in the section can be established for each end of the prism to be estimated.

Except in rare cases, the factor 12 is acceptable for reducing cubic feet to tons for ore in place. Ore that was figured at 12 cu. ft. in place, by test has been found to require 17.5 as a factor when it has been in the stock pile for some time. The factor 12 is virtually standard; if another figure is to be used, it must be determined by test.

At least two estimates should be made, one for the ore actually shown by drilling and the other, a more liberal estimate, including ore in justifiable extensions. In the estimates of ore actually known, the ore is not considered a foot wider nor a foot deeper than what is absolutely shown by the analyses of the ore in the holes drilled; as to length, the standard or regular assumptions are made. That tonnage is absolutely present. In the second estimate personal judgment can be allowed to determine extensions of all boundary lines for the ore as one believes justifiable; an estimate based on such outlines gives an idea of what tonnage may be expected within reasonable limits.

Average Analysis of Ore Deposit

Many extensive calculations have been made to determine the relative accuracy of the different methods used for obtaining an average

analysis. As the samples are taken for every 5 ft. of iron-bearing formation, and as the drill holes are usually numerous, the analyses for iron and phosphorus are sufficiently abundant to obviate any serious error regardless of by what method the average is determined. It has been found that adding the analyses and averaging them is the simplest method and produces an accurate figure. Often silica, alumina, and other elements, are not analyzed as often as are the iron and phosphorus; in such instances one should weight the available percentages by the tonnages represented. However, if analyses are too few, no attempt should be made to average them by any method. Analyses may be few in number and be usable, but then they should represent ore from widely scattered parts of the deposit, or the average figure will indicate the content of only parts of the deposit.

Methods of Sampling and Estimating Lead-silver Ore

As WILL be noted from the paper following, the sampling and estimating of limestone replacement deposits is entirely different from such work on a vein or disseminated ore deposit.

Sampling and Estimating Cordilleran Lead-silver Limestone Replacement Deposits

By Basil Prescott, El Paso, Tex.

(New York Meeting, February, 1924)

In the science of evolution of the species, there is a law which, simply stated, avers that the history of the individual repeats the history of the race. Similarly, if unassisted by the experience of others, each successful engineer would perfect and develop his methods and technique from the rough grab sample up to the carefully taken, recorded and studied samples and observations which now determine and decide the expenditure of the large sums required for modern mining. Hence real progress results from the interchange of experience, as in that way the entire experience of the profession can be made available to every engineer. The literature of this subject is already voluminous, but some progress has been made in the gathering of data and in their compilation or presentation.

More Romance than Developed Ore in Limestone Deposits

In certain types of deposits, methods little known 25 years ago have been developed, perfected and standardized to a degree that lifts sampling from the field of manual labor to the plane of an applied science; for instance, the modern methods of estimating the so-called porphyry coppers. With some types of orebodies, we have become familiar, and sometimes on too easy speaking terms, with millions of tons of ore in reserve and tens of millions of dollars; while with others even these once marvelous figures sink into insignificance before the array of ciphers that are presented. In such company, the average limestone replacement orebody is entirely out of place, for its reserves seldom reach a figure to arrest the attention of the modern engineer, and more often are insignificant. It is only when the records of the production of some of the older camps are studied, which show them to have continuously produced over very extended periods, that a realization is borne home that occasionally this type is worthy of consideration even in these days of gigantic tonnage.

"The reduction of mining to the basis of manufacturing," of which we now hear so much, is certainly not a predominating feature of the working of limestone replacement deposits; and perhaps a little more of that romance and adventure which is supposed to make mining attractive may be retained there than in any other modern mining. Certainly the methods employed in the extraction of the ores, while often picturesque, lag far behind those now in use in other types, whether considered from the standpoint of equipment, system of mining, efficiency per man shift, ore blocked per foot of development or tonnage in sight. Some of these conditions may be inexcusable, but others are inherent to this type of deposit and cannot well be avoided, as will be confirmed by those who with more enthusiasm than experience have tried to lift the operation of such properties to a more modern plane.

Engineer's Opinion a Considerable Factor

The sampling and estimating of limestone replacement deposits is quite a different matter from sampling and estimating a modern vein or disseminated body, even under the most adverse circumstances encountered in those types. In the latter, a well-trained corps of samplers accompanied by mine surveyors and draftsmen can often present data to an engineer from which he can correctly report without personal inspection. In replacement deposits, on the other hand, the personal factor is considerable and the result of sampling and estimation is more often an opinion rendered than a logical presentation of facts with supporting data.

Most sampling and estimation, when carried out on any considerable scale, is for the purpose of valuation, whether for owners or for prospective purchasers; and if a fair valuation of the property is not reached, the examination has been a failure. This desired end must be kept constantly in mind by the engineer who undertakes an estimation of a limestone replacement deposit, as a simple list of samples and blocks, no matter how carefully compiled, will seldom serve to correctly value such a property. In most metal mines, it is usually desirable to estimate the ore in sight in as cold-blooded and mechanical way as possible, keeping free from opinions or ideas that might bias or affect the results, reserving all such for the consideration of the future possibilities. In limestone replacements, there is seldom a single block of ground estimated in which the final result is not seriously tempered or controlled by the engineer's judgment.

Determination of Form of Orebody Essential

The fundamental basis of an estimation of an orebody of this type is a thorough knowledge of the occurrence of the ore, its loci of deposition,

forms it may take, deformation or alteration it may have suffered, and a comprehensive and, if possible, definite idea of its origin and channels of ingress. These factors vary so widely that they must be determined for each deposit; almost for each orebody. In a typical case, the ore is chiefly oxidized, soft and friable, lying enclosed in a hard solid mass of limestone. The variation of form is infinite; the bodies may occur with their major axes parallel to the bedding or these may cut the bedding at strong angles. When parallel to the bedding, they may be anywhere from approximately horizontal to nearly vertical and may be widely distributed throughout the favorable beds or may be confined fairly closely either to the intersection of certain definite fissures with such beds or to definite structural features. When the trend of the ore deposit is across the bedding, the form may occasionally be that of a filled fissure but more often that of a very irregular pipe or chimney, with greater variation in trend and in cross-section than is found in bodies that parallel the bedding.

The workable deposits seldom, if ever, conform to the descriptions that were formerly given for such deposits, which called for disconnected chambers or caverns scattered through the country rock like plums in a pudding, at junctures of certain fissures for limited extents and depths only. These were connected one with another by minute, almost invisible, seams which, in development, had to be followed with extraordinary skill by the ultra-intelligent miner. No such deposits are now producing ore in Mexico and the writer has not been able to obtain conclusive evidence that ore deposits that would answer that description have ever been worked in the Mexican province.

While the replacement orebodies that cross the bedding do not, as a rule, survive a change in the character of the country rock, they often show great continuity, and pipes with a vertical range of 3000 ft. are known; while "mantos" that follow the bedding with twice that length are not exceptional. Although the cross-section will usually vary inversely with the distance from the source, it is at no point necessary to make capital of stringers; and after the body is mined out, the sectional area at the smallest point will be found to be, even in extreme cases, an area that is a large fraction of the average or maximum area of cross-sections for that orebody, up to that point. The actual average sectional area of different pipes or mantos may vary from a minimum of 2 sq. m. to a maximum of perhaps 200 sq. m. Where the orebodies follow the bedding and where the bedding is nearly vertical, the ore distribution may resemble the oreshoots along a vein and the cross-section vary accordingly; but where the beds are more nearly horizontal, the ore commonly occurs with the height either a considerable factor or a small multiple of the width.

Previous Production Most Important

Having determined the characteristic form or shape and occurrence of the orebodies, a careful and thorough study of previous production as to tonnage, grade and analysis should be made from production and shipment records; and, where it is possible to do so, this should be compared with definite stoped section of the mine, to determine as accurately as possible the tonnage per linear foot of orebody in the case of flat-lying bodies. These comparisons are of great assistance as checks against the estimated tonnages of orebodies yet unstoped, and while always valuable in any estimation of mining tonnages, are most essential in this type and often should be given more weight than the insufficient measurements of the ore itself that are usually available. It is often a temptation to slight this part of the work, as a conscientious engineer seldom cares to devote sufficient time to the stoped-out portions of the mine under consideration; but his principals will receive no greater return from any part of the examination than from the careful comparison of ore produced with the stopes from which it came. Fortunately, in many limestone replacements, the walls stand indefinitely, and were the records of tonnage and grade even a fraction as perfect as those of volume, a considerable degree of accuracy might be obtained.

Correction for Included Limestone

In many limestone replacement deposits, both in oxides and sulfides, a large amount of barren limestone is included within the walls of the orebody. A characteristic of replacement deposits is the extremely sharp line of demarcation between the ore and the country rock; disseminations are most uncommon and the grading of the values into the walls is found only in rare instances. Consequently, the included country rock is not usually considered as a part of the ore and is neither sampled nor taken into the estimates; and when the body is mined, it is thrown to one side. There is often found on top of a flat-lying oxidized body, a mass of broken limestone which can be readily excluded from the tonnage though it must appear in the cost of extraction; but where the limestone occurs scattered throughout the mass of the ore it is impossible to keep this from inclusion in the estimated blocks, which are then best adjusted by applying a factor determined from previously stoped areas or by the inspection of a sufficient number of exposures. Such inclusions are often considerable and seldom negligible, varying to a maximum of perhaps 30 per cent. of the material within the stope walls, the variation depending chiefly on the character of the limestone, the shape of the orebody, its relation to the dip of the strata and similar features. In many cases, the factor is fairly constant for an entire body or even for a district, thus lessening the work entailed in its determination.

Specific-gravity Tests

A larger percentage of limestone replacement ores is oxidized than in any other type of metal mines, and in some cases only the oxides are commercial. These oxides are usually porous, loose, often friable and prevalently cut by open spaces. Probably the most fruitful source of error leading to an overestimation of oxidized ores of this type in bodies of fairly well established outlines is in the assumption of the specific gravity or ratio of tonnage to volume; and certainly in no other type can such an assumption vary so far from the actual relation. At Santa Eulalia, Mexico, it was found than an average sulfide ore with a theoretical specific gravity of 4.8 had a specific gravity in masses of about 4.2, while the oxidized ore resulting from the same had a theoretical specific gravity of 4.3 and an actual specific gravity of 2.4, giving a porosity of about 12 per cent. for the sulfide and of about 44 per cent. for the oxides. These are rather remarkable but by no means extreme, and show the necessity of determining the factor before attempting to calculate tonnage. For rough work and as a check against determinations, a good rule is that the oxides, where soft and friable, will have a little higher density in place than when broken down or in the mine cars. The determination of the specific gravity, including porosity, of a soft friable loose mass in place requires some ingenuity and care, but if the tonnage is worth sampling and estimating, this factor is well worth determining for several distinct representative classes of oxides. If mining is being carried on, an approximation can be made by measuring up, extracting and weighing a small block; otherwise it is advisable to carry out a series of tests on lump ores and fines for the determination.

Normal Grade of Ore

Every replacement orebody has a certain "normal grade" of ore which might be defined as the average of all ores above that critical point where the grade commences to drop very suddenly to traces only. The sharp demarcation in the outlines of the orebodies and the absence of grading off of mineralization into the country rock are among the most pronounced characteristics of replacement deposits; and in the same way, there is nearly always a definitely establishable point below which the tonnage for each unit drop in grade is insignificant until the realm of traces is reached, while the tonnage for each unit rise above that point is considerable until the maximum is approached. This critical point and the resulting normal grade are independent of commercial considerations, costs, metal values, prices, etc., and should not be confused with the economic limit or lowest grade which such commercial considerations permit being handled for a suitable profit. In many types of ore deposits,

the tonnage increases inversely as the grade; and an improvement in costs, metal market, treatment charges, etc., immediately results in a large increase in tonnage available for production. This is not usually true with replacement deposits provided that the normal grade is equal to or better than the cost of production plus the required profit. This is the first point to be determined by sampling and it is an essential one, for the difficulties of operation are greatly increased and the possibilities sadly cut if it is necessary to practice selective mining in order to keep the grade of the product above the normal grade. On the other hand, nothing is gained by any attempt to lower costs by increasing the tonnage through the production of ore lower than "normal"—usually such low-grade ores in oxidized replacement deposits consist of the iron or gypsum leachings which carry little value, but sometimes are of good grade. It is essential, therefore, that careful sampling determine these points for both tonnage estimation and operation, or the product will drop below normal grade, a condition that may not be fatal to operations provided there is still a margin of profit, but one that points to poor management and lack of close supervision.

The normal grade cannot be accurately determined from shipment records as, for example, a mine may have for years produced a lower grade than the normal. In many cases, owing to the suddenness of the drop in grade below the critical point, the total tonnage above the lowest grade that will pay its own way and the total tonnage of normal grade are not very different, but careless mining may have vitiated the results obtainable from shipment records. The normal grade can best be determined by careful sampling for that specific purpose.

Location of Samples

In silver-lead deposits of the flat-lying or "manto" type, prospecting is often confined as much as possible to the horizon next the back where shrinkages, or open spaces caused during oxidation, are followed or expected, greatly expediting the opening up of the orebody when they are encountered. Care should be used in accepting and including samples taken under such conditions, in the general average, for this upper part of the orebody always contains higher values than the underlying ore. It seems probable that the oxidized solutions heavily charged with active sulfates, coming in contact with the clean limestone of the back, react and deposit their burden on the surface of the oxidizing mass below, thus forming the shrinkage by solution of the limestone and the heavy band of lead carbonate so prevalent at that point by precipitation. On the other hand, raises put up to such a body from below are also untrustworthy in many cases, for as a rule such raises are located at those points where oxidizing solutions have carried down to the horizon of the develop-

ment or extraction drifts leachings from the body along some open fracture, and an erroneous idea of both the thickness of the body and its grade results. Where faces are exposed in breast stoping, careful sampling and observation will often give information and data that may be used with greater confidence over a large area than local samples or measurements obtained throughout the area. Lacking such a breast, winzes which expose a section from roof to floor should be given great weight, while shallow pits should be avoided. It is sometimes possible to cut hundreds of samples at ordinary sampling intervals in the exploration drifts and crosscuts of an orebody, the returns of which may then be averaged and will give a very satisfactory result, but only for that vertical height which the drifts expose. As these resultant averages must then be used as units with the comparatively few samples representing other horizons taken from such winzes and raises as may be available for sampling, it is often a question whether extreme care or close sampling intervals on the levels are justified.

In an orebody of the chimney type, the level samples are of more importance, especially if the winze and raise samples show the absence of horizontal zoning of the values.

Danger of Overestimation or Underestimation

If the sampling is directed toward obtaining the average and normal grades of each orebody rather than the average of any particular block that happens to be in sight or the average of all the samples that can possibly be taken in the development work open, there is no great danger that the engineer's estimate of grade will be far wrong; but the estimate of tonnage that may be produced from a given area is seldom so close an approximation to the final result of actual stoping operations as those made in other types of ore deposits. Almost without exception, the actual known orebodies are overestimated in the oxides and very often are underestimated in the sulfides, and in many instances the discrepancies are most remarkable, particularly in the case of the oxidized bodies. Often it is only after a few thousand tons have been mined and the appalling size of the hole in the orebody is seen that a true realization of conditions is reached. Fortunately, in many instances, irregular extensions are found beyond calculated limits that tend to counter-balance the discrepancies, and new bodies are discovered that save the operation, but usually these result from a very active campaign that is inaugurated after a knowledge of the situation is gained.

A number of the most important points that must be kept constantly in mind, to avoid this result, have been mentioned; *i. e.*, (1) a clear and definite idea of the process of the primary deposition of the particular orebody, and of the secondary processes during oxidation; (2) a careful comparison of the shipment record with the volume removed; (3) a

fair allowance for the included limestone which will not be shipped; (4) an accurate determination of the specific gravity of the mass and its percentage of open space. More important than these, however, is the determination of actual cross-sections of the orebody. An oxidized silver-lead replacement orebody of considerable size may be prepared for examination in such a way that overestimation can scarcely be avoided, but it may also be laid out so that its estimation will be comparatively easy. The important point is the determination of the entire cross-section at well-spaced intervals rather than any attempt to cut the entire body into blocks. A much smaller amount of development in ore confined to a definite plane at right angles to the trend of the orebody and thoroughly outlining the walls of the deposit at that point, with a single connecting winze, raise or drift in ore between the successive cross-sections, will give better data than a much greater amount of work scattered throughout the orebody. The spacing of these planes of concentrated effort should be 100 to 150 ft. in steeply inclined chimneys; but may be two or three times that distance in flat-lying bodies of the manto type. As mentioned, it is extremely desirable to determine the average tonnage that may be expected per linear foot rather than the tonnage of a given block, for that factor greatly assists in estimating the future possibilities—and if the examination is being made for valuation, the possibilities are often more important than the tonnage in sight.

On a vein, the future tonnage will generally come from additional levels or lateral extensions of the known mineralization; in a replacement deposit, it comes very largely from the finding of new orebodies which may not even be indicated at the time of examination. It is, in that case, impossible to give more of an estimate of such futures than the simple statement as to the favorableness or unfavorableness of the area for other orebodies. The problem is strictly a geologic one and should be kept distinct from the sampling and estimation of the ore itself.

Cutting Samples in Oxide Ore

The actual cutting of the samples in oxides usually presents little difficulty and no new features. Care is taken to keep the channels even and at right angles to the banding, and it is considered good practice to take a much larger sample than that ordinarily taken in other types in order that the effect of the occasional hard lump that falls into the canvas or into the box may not be out of proportion with the whole. Most of the labor is involved in quartering and reducing to a size suitable for sending to the assay office.

In general, it is not worth while to expend much effort on the sampling of the walls of stopes already mined even though these are apparently quite attractive. Under the system commonly in use in extraction, very often the stopes are carried along quite rapidly to take out the main

mass of the ore, and are later reworked and thoroughly cleaned. It is surprising how much tonnage often results from this second operation, but there is no way to mechanically estimate the probable tonnage or grade that can be won during the process. An experienced man can sometimes make a rough approximation, but careful sampling and measurements do little toward assisting him, and the cost, as compared with the benefit, is relatively high.

A silver-lead replacement prospect should always be treated with a certain amount of respect by the examining engineer, as its potentialities are not always commensurate with its appearance; and if that respect lead to a refusal to pass judgment on such properties, then only is he fairly sure of making no serious error. Unfortunately, however, it is not always possible or expedient to refuse such commissions and the engineer must then render an opinion from the data he can acquire. Sampling should consist of scattered and irregularly spaced cuts aimed at the determination of the ratio of the precious metals to the base metals, and the grade of the ore that may be expected from the deposit rather than the grade of the material exposed. This ratio and grade cannot always be determined by sampling limited exposures, but a decision can at least be made as to whether mining, transportation, treatment charges, etc., can be met if larger bodies of the same content are encountered. The prediction of the future of the prospect is often an extremely difficult geologic problem.

Sampling Sulfide Ore by Drillholes

The sampling and estimation of the sulfide bodies in replacement deposits present a quite different set of conditions from those encountered in the oxides. The ores are often massive, sometimes fairly uniform in shape and in distribution of values, but more often exhibit great irregularity. In the latter case, the sulfide occurs in bunches, bands, fissures and so forth between and including much unreplaced limestone as layers or blocks, and it is not uncommon for faces within the later determined limits of an orebody to show practically only barren limestone. The metallic content is also very often irregularly distributed; a layer or band of one sulfide to the practical exclusion of all others is not unusual and these factors not only render sampling difficult and accurate estimation almost impossible, but seriously interfere with prospecting and development.

If the orebody has been prepared for stoping, channel samples intelligently laid out rather than mechanically spaced will give good results; but in many instances, better results have been obtained by drilling the ground with machine drills and using the cuttings for a sample. It is not difficult to devise methods and apparatus which will prevent losses through dusting, and the errors introduced are slight. The size of a

starter is always slightly larger than the drill which follows, and where extreme accuracy is required or it is felt that an appreciable error might thus be introduced, it might be necessary to make allowances for this factor; but in the majority of instances, no such refinements are warranted.

It is sometimes the practice in sulfides, as is standard with oxides, to reject portions of the limestone and lower the tonnage by the amount that may be thrown out rather than to lower the grade; but this must be very carefully done, and only the larger masses that can be readily handled in breaking operations taken into account.

Where fairly uniform and free from banding, short vertical holes of from 6 to 12 in. deep, spaced 12 to 18 in. apart along the line of the sample, have given excellent results; and where the ore is hard and massive, require less effort and expense than a channel. A certain number of holes covering a determined sampling length are taken as a single example; and, to give good results, the line of these holes must be carefully laid out before commencing operations and the individual holes spotted by careful measurements of the predetermined interval, irrespective of the condition of the back and ease of collaring the holes.

Where horizontal banding is suspected, and it is quite prevalent in this type of deposit, channels across a stope back give very misleading information and even along the walls are quite unreliable. For such cases, vertical drill holes a meter or more deep, spaced at intervalsabout equal to the depth along the line which the channel would have followed, and these lines of holes spaced as the channels would have been spaced, will yield excellent results.

When vertical banding is present, the holes may be put in at angles of approximately 45° and so spaced that one hole bottoms in the band in which the next one collars, resulting in the equivalent of a hole clear across the orebody. This method has checked against shipments within very satisfactory limits whereas channel samples in the same stopes, owing to the hardness of the ore and irregularity of the metal content, varied beyond the desired accuracy. A three-legged horse provides a convenient rest for the tail piece of the ordinary stoper drill, and one that is quickly adjusted to any length of steel. The drill passes through the side of the bag used as a dust collector, a piece of gasket rubber on the drill steel inside the bag prevents dust from following the steel out through the hole in the canvas, and a Jones sampler aids in cutting the product to the required size. Two men are needed, one on the drill and the other to hold the bag against the face. The bag is hung on a heavy wire hoop and this is held against the face with a light forked iron rod in the hands of the assistant. Innumerable variations can be imagined.

The ore is usually bedded or banded and there is often considerable difficulty in getting the samples so scattered that they represent the area under consideration. Careful mining often yields a better result

from the sulfides than the sampling warranted, especially if a certain amount of selection and sorting is possible. Such bodies are often underestimated, simply because the ultimate limits are not exposed and apparent walls are accepted.

Consideration of Future Possibilities

When all the data are in and the mine or deposit under consideration has been carefully mapped, sampled, estimated and technically weighed, and the profits on the ore in sight have been painstakingly determined from such data, then the entire accumulation should be carefully filed away for reference and the future possibilities given the consideration due them. Almost invariably, this is the deciding factor and it is often one where a well guided imagination, backed and assisted by business experience, will come nearer to the correct result than the best technical effort.

Deep-hole Prospecting at the Chief Consolidated Mines

By Chas. A. Dobbel, Eureka, Utah

(Salt Lake City Meeting, September, 1925)

The Chief Consolidated properties are situated in the Tintic mining district of Utah, being included in Juab and Utah Counties, about 70 miles south of Salt Lake City. The drilling referred to in this paper has been carried on in the Chief Consolidated No. 1 mine, the Gemini mine, and the Eureka Hill mine at Eureka, the Grand Central mine, above Mammoth, and the Apex Standard mine just south of Dividend.

The occurrence of ore through the district is typically in lenses on bedding or breaks, and in connecting pipes, without accurate geometrical trend so that the ratio between the total length of development in drifts and crosscuts to the square feet of ore exposed is very high. A section of one of the western channels (Fig. 1) through the Tintic district, shows, roughly, the irregularities calling for excessive development work, so typical of the Tintic mines. This irregular trend of the ore constantly calls for excessive cutting up of the whole area for offshoots and the connecting links always present. In the planning of prospecting and development, no regular system may be laid out without overlooking some orebodies or segments of orebodies.

It was with these facts in mind that Cecil Fitch, the general manager of the Chief Consolidated Mining Co., decided that some method of drilling was the more feasible way of successfully cutting up and bringing about the discovery of ore believed to exist; yet this drilling had to compete with a great margin of cost over crosscutting, as a crosscut in the Tintic district, even though barren, brings out geologic facts that, in many cases, are worth the price of the working itself. Diamond drills met with the success that might be expected in a district that has a reputation for the wear and loss of diamonds; after 6 or 8 months, this method was discarded on account of the high costs. Simultaneously, a small churn drill outfit was set up underground but a few shifts proved its inapplicability. At the same time, the ordinary hammer drill was used to drive prospect holes with jointed steel.

Just what equipment to develop for this work with the hammer drill involved much experimenting, as nothing was found in the literature

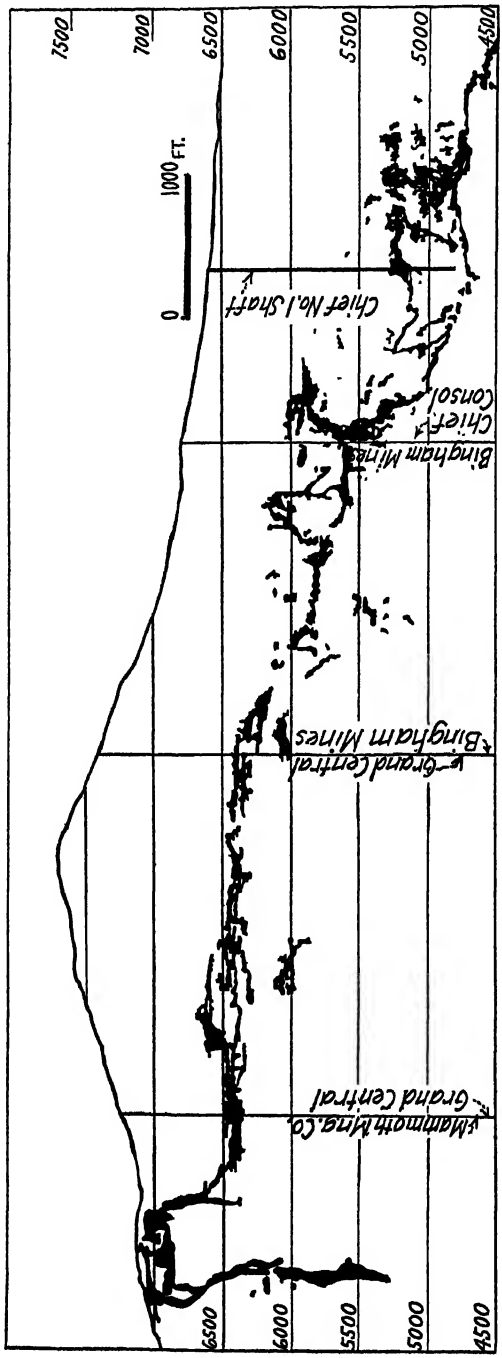

FIG. 1.—SECTION OF A WESTERN ORE CHANNEL THROUGH THE TINTIC DISTRICT.

pertaining to similar attempts at drilling. Forerunners of the method were the Elm Orlu Mining Co., of Butte, Mont. when unwatering a shaft from an underground working by the use of extra heavy pipe with the regular bit and shank on the ends; the use of the hammer drill by the Anaconda Mining Co. in drilling through old gob of its fire zone with steel made up of diamond drill rod; and the experiments of the North Butte Mining Co. along similar lines for prospecting purposes. The latter company, when it suspended operations, had developed a steel of this extension type of 1¼-in. hollow round, with upset shoulders of 1¾-in. diameter, with a taper thread that had been used successfully to cover the prospecting of the altered areas of their veins for distances of 40 to 50 ft. Some of this steel was shipped to the Chief Consolidated for experimental work. As this steel proved successful only for depths of its original design, and the need of a stronger joint was found, the work on this development was started with three objects in view: (1) to produce a joint that would stand up with the drilling; (2) to produce simplicity in the mechanical work in making the joint; (3) to produce a joint that would be easy to manipulate underground and not gather dirt and particles, to which most joints are so easily susceptible to the detriment of the threads. A drill manufacturing company[1] through the suggestions of Howard R. Drullard and E. F. Terry brought out the joint we are using today; it is shown in Fig. 2.

The primary feature of this joint, which imparts strength to the splice, is that the ends of the steel butt together in practically their full diameter, hence the blow is entirely on the steel itself and not on the threads, as in previous designs. The strain in the sleeve is torsional and tensional; the compressive force is entirely in the steel.

Simplicity of the joint is obtained by threading the steel by forging; this is done on the ordinary drill sharpener, equipped with suitable dies. It requires but a few minutes to forge threads on both ends of the steel section, after which the ends are squared in a lathe, and then tempered. The sleeve was originally tapped from high-grade steel, but the high price of this operation could not compete with cast sleeves, made from one of these carefully tapped sleeves as a pattern, which have been successfully used for over a year and a half at the Chief Consolidated.

Simplicity in operation is brought out by reducing wrench work to a minimum. A light tap with the chuck wrench has proved to be sufficient to loosen this joint after use, and the rotation of the machine takes care of the joint to insure the proper taking up for drilling. A reverse button on the rotation of the machine also aids in the unscrewing of the joints. Dirt does not lodge in the threads as the forging gives a rounded,

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[1] The Denver Rock Drill Mfg. Co., Denver, Colo., whose helpful assistance in the development of successful equipment for this work deserves full acknowledgment.

coarse thread (a double thread of ½ in. pitch) for which fine particles have no affinity.

The development of the steel was the first step toward successful prospect drilling with the hammer drill but a factor of equal importance

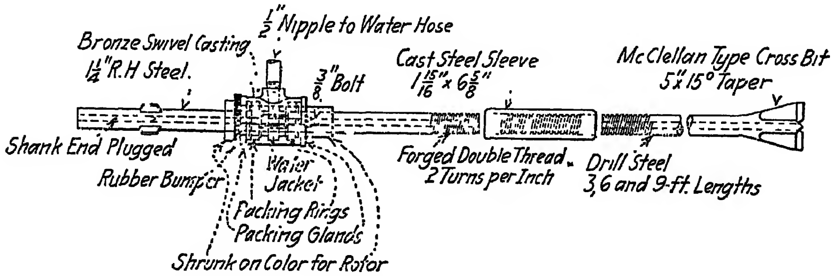

Fig. 2.—Swivel and steel assembly for deep-hole drilling with hammer drills.

was the type of machines used. Almost all standard drifters were used in the experiments, but the point of prime importance was the rotation, as the heavy weight of the steel was adding an excessive duty to the

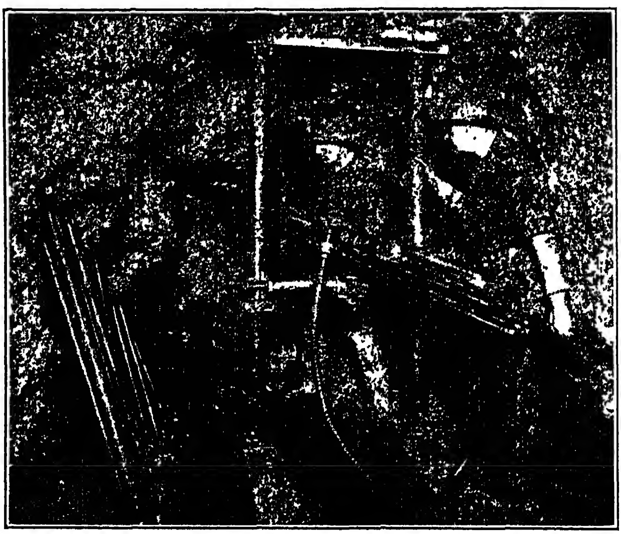

Fig. 3.—Heavy equipment for drilling deep holes at the Chief Consolidated; set-up of heavy drifter drilling a hole 227 ft. deep; this machine has a depth record of 272 ft.

machine. The machine finally adopted was the independently rotated type; in over a year's operation not one part was replaced in these machines, though we had drilled past the 270-ft. mark, and some machines are now running close to two years with the original parts.

The bits are an important factor, especially where a hole of some depth is depended on. The McClellan type has been taken as standard (Fig. 2) for both the 1-in. and the 1¼-in. round steel. With the former, the starter is 2½ in.; and with the latter it is 3⅛ in. in diameter. Changes are made with ⅛-in. drops, with eight changes for both sizes before the size of hole gets too small for the outside diameter of the sleeve on the joints. Bits of this type have required changing after drilling as little as 6 in., while at other times we have drilled as much as 67 ft. with the first bit in the hole. The life of a bit varies directly as the ground drilled.

Overcoming too quick a loss of gage is brought about by two methods. The first and most desirable is reaming with bits of the same original size

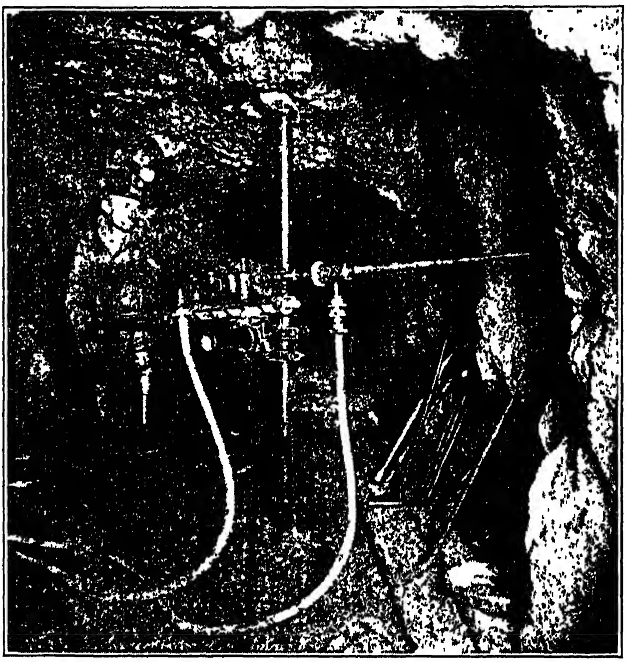

Fig. 4.—Light drifter drilling a 100–150 ft. hole at Eureka Hill mine of the Chief Consolidated; this type machine has a depth record of 200 ft.

of the one taken out. This is the standard method at the Chief Consolidated and in some holes six to eight bits of the same size have been used before the drop to the next size was taken. The other method is to place a number of feet of smaller diameter steel ahead of the standard 1¼ in. steel, but this has not always proved satisfactory as there is considerable vibration of this steel before it transmits a blow to the face of the hole.

The set up evolved is shown in Fig. 3. The original plan was to set up the ordinary column and cross arm, but the alignment of the hole is

such an important consideration when drilling to depth that the two columns and the double-handed cross-arm were necessary to keep the machine rigid and prevent the turning of the cross-arm on the column.

Water is introduced into the steel through a swivel joint on the shank of the steel; its construction is shown in Fig. 2. Through a long length of steel, excess pressure is important; as the entire underground workings of the Chief Consolidated mines are accessible to a piped water supply this pressure is easily obtained except in remote places. For places of low pressures, a small duplex pump, about say 3 by 2 by 3 in. giving a theoretical pressure of 150 lb. with 100 lb. air pressure is used. This pump costs between $50 and $60 and is of great value around a deep-hole drill outfit for emergencies. The pump has been used to advantage where the water supply was limited and had to be taken into the workings in barrels. In this instance, the water was allowed to settle in the workings and was then used again by allowing the drain of the hole to act as a sump for the pump. In the drilling of down holes the swivel has the added feature of taking water from three-fourths to five-sixths of a revolution and then air for the remainder of the turn. This forms an air lift in the hole and clears the hole of cuttings readily, even to depths of 100 ft., which has been our deepest requirement to date on vertically down holes. To study this method, a system of valves were hand-operated intermittently until it was found feasible to make the rotation of the steel accomplish this valve opening.

In Figs. 2, 3, and 4 are shown all the necessary equipment used at the Chief Consolidated; the only change in the machine is the sealing of the water and air tubes. The heavy equipment (Fig. 3) using a standard independently rotated tunnel machine and drilling with 1¼ steel is the outfit assigned to the drilling on contract. The lighter equipment (Fig. 4) is used by the independent leasee, who borrows the equipment from the company to prospect his territory. This consists of a lightweight, independently rotated drifter (mounted sinker) and 1-in., round, hollow steel. It may generally be said that the heavy equipment is used for holes from 100 to over 200 ft. in depth only, and the lighter equipment for holes up to approximately 150 ft. The light equipment has actually drilled 200 ft., without noticeable excess strain on the machine.

Methods of Sampling

Many methods of sampling the drill holes, including filtering, were devised and special boxes were made and tried, but the ideas were too complicated for the average miner to handle, were cumbersome, and had no advantage over the method of placing a powder box at the mouth of the hole to catch all the cuttings. These boxes are changed for each 3-ft. length of steel added so that a continuous sample results at 3-ft.

intervals. All the cuttings for the 3 ft. are sacked and sent to the examination room for quartering and examination, whence they are either discarded or sent to the assayer. The type or formation of limestone for each sample is determined by physical examination, so that the limes drilled through are known. In fact, these cuttings are treated much the same as diamond-drill core and, in the Tintic district, have proved of as much value.

It is admitted that there is a loss of slimes in this method of taking a sample but, for some reason, resampling along the hole after it is opened up checks with the original sample. There have been quite a few cases that have caused a discussion on checks in sampling, but I do not believe there is anything clear to which a theory may be applied except the possibility that the slimes carry the same values as the cuttings in this locality. This was checked up on the 1500-ft. level of the Chief No. 1 mine and found to be true; where the cuttings were running from 4.8 to 5.2 oz. of silver from a hole, the slimes being run to waste at the same time ran from 4.5 to 5.0 oz. but this is the only actual case of checking up that can be cited.

Uses of Deep-hole Hammer Drill

The primary use of the deep-hole hammer drill is the finding of new ore or the locating of indications that might open up to ore when explored; its use, however, is by no means limited to this alone. This drilling has been conducted with such low cost that it forms a very cheap and desirable method for the disproving of ground, as well as proving the extension

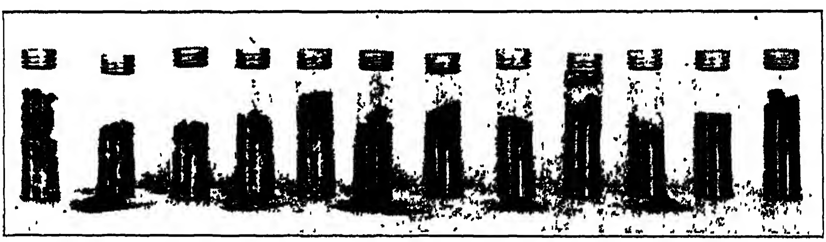

Fig. 5.—Parts of twelve samples from a drill hole starting in limestone and entering a mineralized zone.

of ore. This means that ground theoretically favorable can be proved without the expense of crosscut. The argument that a crosscut is needed anyway if ore is struck is offset by the fact that the drill hole points the direct way without random crosscutting. Then, too, there are cases where the hole has drained the orebody of water by the time the development crosscut reached it so that the cost of mining was materially reduced.

A recent development in the use of drill holes has been the study of the cuttings, which actually puts a drill hole on a competing basis with a crosscut as far as geological information is concerned. This not only locates the "good" and "bad" limestones (those susceptible or not as susceptible to the deposition of ore), but from the cuttings the limestone horizons can actually be mapped, which extends the geological knowledge beyond the actual workings of the mine. The location of faults, or breaks, that may be mineral bearers can also be located by this study of cuttings or changes in the cuttings. Portions of twelve samples from a hole starting in limestone, with a decided change to a mineralized zone, are shown in Fig. 5. Physical changes in the limestone horizon are not shown as clearly in a photograph, but their characteristics and keys can be noted readily on examination. These samples give some idea of the size of particle called the "core," as well as the well-defined changes that take place from sample to sample.

Drainage advantages are also to be noted. In the Chief Consolidated, the drainage of the lower levels has been nearly centralized with the aid of drill holes and, in some instances, excessively wet areas have been drilled and allowed to stand and drain until work could be carried on with less inconvenience. There is also the readiness with which a shaft or winze may be unwatered ahead of the crosscut planned to hole through.

Among the miscellaneous uses of the deep-hole hammer drill might be noted the case where the transformers for 1940-ft. winze were placed 350 ft. away from the 1940-ft. hoist. To eliminate the copper loss on the line leading to the hoist, a hole was drilled through to the motor for the cable; the distance was then but 50 ft. In the case of difficult holing-through problems with raises or drifts, the proposed location can be tested with a drill hole to insure correct spotting, and the hole then can be the guide line of the final working.

Costs of Deep-hole Drilling

The progress of contract drilling in the No. 1 mine from October, 1923, to April, 1925, as well as interesting averages hinging directly on costs, is shown in Table 1.

TABLE 1

Month	Total Depth Drilled, Feet	Total Drill Shifts	Advance per Drill Shift, Feet	Man-shifts	Labor[a] Cost Per Foot	Average Depth of Hole, Feet	Deepest Hole for Month, Feet
1923							
Oct.	233	13.5	17.3	27	$0.58	39	48
Nov.	599	26.5	22.6	53	0.44	54	94
Dec.	742	61.0	12.2	122	0.81	57	227
1924							
Jan.	1618	111.0	14.5	222.0	0.69	54	170
Feb.	2148	113.7	18.9	227.5	0.53	89	272
March	1932	91.3	21.2	182.5	0.47	102	201
April	1349	94.5	14.3	189.0	0.71	71	187
May	947	54.0	17.5	108.0	0.57	73	150
June	430	34.5	12.5	79.0	0.80	62	123
July	2091	82.0	25.5	172.0	0.41	99	177
Aug.	2715	93.0	29.2	194.5	0.36	102	177
Sept.	2852	96.5	29.6	196.0	0.36	95	186
Oct.	3052	97.8	31.3	211.5	0.35	93	189
Nov.	3071	90.5	34.0	180.5	0.28	93	192
Dec.	2748	96.0	28.6	209.0	0.76	96	177
1925							
Jan.	2431	94.5	25.7[b]	203.5	0.81	94	215
Feb.	1981	90.0	21.8	193.5	0.90	87	196
March	2872	99.7	28.6	210.0	0.79	85	168
April	2421	98.0	24.7	199.5	0.81	84	171

[a] This labor cost is figured on a basis of $5.25 for the driller and $4.75 for the helper per shift.

[b] The smaller advance per machine shift, hence the increased labor cost per foot, in January, February, March, and April, 1925, is the result of the experimental work on down holes (vertically down) that had been in progress.

The total cost of drilling, including all items, is shown in Table 2. The depreciation of equipment was arbitrarily set to be charged off in two years, sleeves and steel were taken at 10 cents per foot drilled, and other charges are direct distributions of costs and cover drilling for the year of 1924 (labor cost here is figured at $5.25 for the driller and $4.75 for the helper).

TABLE 2

	Cost per Foot
Labor (driller and helper)	$0.44
Miscellaneous labor and supplies	0.13
Air charge and steel sharpening, including making up new steel	0.11
Depreciation of equipment	0.17
Company supervision	0.07
Blasting out for set-up room	0.05
Total cost per foot	$0.97

These figures include all experimental work on the developing of the outfit. The machine-shop costs are grouped under Miscellaneous Labor and Supplies. As the advance in the development of the equipment and the largest part of the experimental work are completed, no doubt the costs for 1925, computed on the same basis, will be as low as $0.75 per ft. drilled.

To drill 22 ft. per day means that the driller and helper at $5.25 and $4.75 respectively, give us the cost of $0.44 for labor. When an average of from 25 to 30 ft. per shift for the month has been maintained, both men receive $1 extra per shift. This is the basis of the bonus plan; for each additional 5 ft. per shift average for the month, an extra dollar per shift is paid. This means that the extra 3 to 8 ft. per day above 22 ft. cost us from 25 to 66 cents for labor. From 9 to 13 ft. extra above the 22-ft. average cost us 30 to 44 cents per ft., etc.

Where lessees are allowed to drill, a sliding scale is arranged figured on the depth of hole; this is footage allowed to partly cover the expenses of the lessee when prospecting for ore. The lighter equipment is used so that one man can handle the drilling in this case; the lessee, however, usually prefers to hire a helper and his average progress under these conditions is about 20 ft. per shift. The average depth of hole is 60 ft., as this drilling is done to prospect the ground assigned to the lessee, which is usually a 100-ft. cube and he starts somewhere within his block. This means that he drills his 60-ft. hole in three days and is allowed $18 for footage—a labor cost of 30 cents per foot to the company, and about a 60 per cent. return to the lessee and partner for their time. This sliding scale is as follows:

For holes 20 ft. in depth	$3.50
For each foot from 20 to 30 ft.	0.25
For each foot from 30 to 40 ft.	0.325
For each foot from 40 to 50 ft.	0.40
For each foot from 50 to 60 ft.	0.475
For each foot from 60 to 70 ft.	0.55
For each foot from 70 to 80 ft.	0.625
For each foot from 80 to 90 ft.	0.70
For each foot from 90 to 100 ft	0.75
For each foot over 100 ft.	0.75

No holes are paid for which are less than 20 ft. in depth.

The other charges against this drilling (the above covering labor charge only) are about two-thirds of the costs of the heavy equipment. This means charges on depreciation, air, steel, supervision, etc.; there is no charge for blasting out for set-ups on lessee drilling, as this is usually done by the lessee on his own time, and the supplies are charged against him by the company.

Factors Affecting Progress

The first factor affecting the progress of drilling is the pitch of hole. The best drilling holes are from 20° to 30° above the horizontal. Holes from 0° to 20° are sometimes difficult to clean, especially if the water pressure is low. In holes from 30° to 45°, the steel handling takes more time; for when the hole grows with depth the changing of bits becomes a problem of pushing the steel back in the hole. Above 45° there is not only the disadvantage of the weight of steel during handling, but on the machine as well. In holes above 60°, it is always desirable to use a counterweight arrangement; a most simple one is a carbide can, rope, and pulley, so that a rock may be placed in the can for each length of steel added. At present, it is only in special cases that a counterweight is used in holes less than 100 ft. in depth. From the standpoint of safety, however, there should be a clamp of some kind on the steel when drilling in steep holes, for if the steel should break above the swivel the driller or helper may be injured if the steel should drop past the machine. When drilling holes downward, progress is affected by the weight of the steel. A bit resting on the bottom of a downward hole, with a few hundred pounds of steel above it, plugs easily and means more pulling of the steel unless the bit is kept well up from the bottom during all manipulation of the steel.

The second factor that affects the progress of drilling is the geological conditions. Loose broken ground means caves in the holes; these holes must be respotted, to avoid this area, or cemented, which takes time and adds to the cost. Open breaks mean loss of water, hence no sample returns to the collar; in a limestone area, as the Tintic district, such open breaks practically mean loss of the hole. The hardness of the ground has been discussed in connection with drill bits; this not only means loss of gage but slower drilling speeds.

The introduction of the bonus system has been a big factor in progress. Table 1 shows a decided increase in progress per machine shift for the month of July, 1924, when the system was introduced. The bonus system has lowered the actual labor cost per foot, but it has the disadvantage of reducing the time a man will spend for the recovery of broken steel or on the slower drilling rock. In some cases, a driller prefers to say that he has lost a hole so that he can take a chance on the next being a little softer drilling; for this reason a close check must be kept on the hole's progress.

Fishing Tools for Broken or Lost Steel

Naturally steel breaks from fatigue. For a long time a weak joint was placed outside of the hole so that if a blow was dealt with too much air, the steel would break at the swivel rather than in the hole, but the development of fishing tools has made this joint unnecessary. In Fig. 6

is shown the "grabber," as it is known around the workings. This is simply a sleeve with corrugations on the inner side, which catch the broken end of the steel; the other end is threaded for attachment to a 1-in. pipe. This tool is also used for the recovery of stuck steel; for instance, when a bit is plugged and sticks so that it cannot be moved. By unscrewing the steel piece by piece, the only section lost will be the bit itself, so that no actual loss of steel is suffered. To raise the end of the steel from the bottom of the hole, a spoon-shaped lip has been placed on the grabber. When the tool is introduced into the hole the lip is kept toward the top of the hole and passes over the end of the steel. The pipe is then turned 180° and the lip forces its way under the broken end, raising it so that it will enter the grabber which is driven on and made fast.

Cost Comparison with Diamond Drilling

The costs, about $1 per ft., cover all costs assigned to deep-hole hammer drilling; these were compared with the work of the diamond

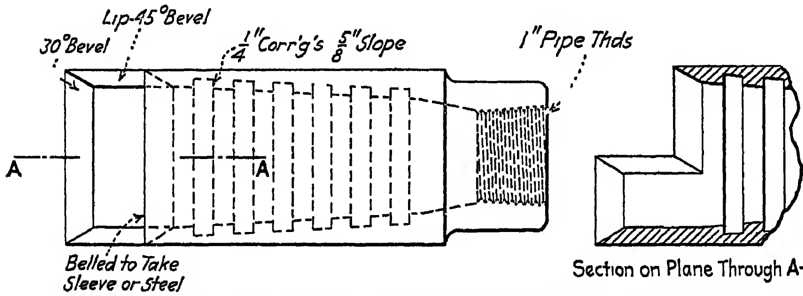

Fig. 6.—Standard lipped steel grabber for fishing for broken or stuck steel.

drill for the 8 months the diamond drills were working in the same formations and against the same conditions at the same time. Over this period, diamond drilling cost, on the average, $4.94 per ft., which I believe is the lowest priced diamond drilling conducted in the Tintic district to date. While the diamond-drill holes were deeper, figured back on the same basis of depth of hole, even giving the diamond drill the advantage of cost, the conclusion is that for the first 250 ft. of any hole, the hammer drill with the outfit here described competes on the favorable side of a 5 to 1 ratio.

Records of Data on Hammer-drill Prospect Holes

The forms shown in Fig. 7 explain themselves. They are printed on both sides of a 5 by 8-in. card and are kept as a file of drill-hole records. The Drillers Instruction form, the Drillers Daily Report, and the Petrographical Report are not shown, as the individual reports are more or less modeled after the standard diamond-drill forms in general use.

Hole No......... **ENGINEERING DATA** ....Mine

PROSPECT DRILL HOLE RECORD

Location:.................................. ....

Level:.........; Block:.........; .... Ft. ....of Sta./Post No...... Elev. of Collar.......

Course:.......; Pitch:..........; Depth:. ....

ASSAY RECORD — GEOLOGIC RECORD

Sample No.	Oz. Au	Oz. Ag	% Pb	% Cu	% Zn	% Insol		Formation, etc.

Remarks:...

Showing Followed.......................Result..... ....

Hole Suggested.............19..

By.......................

Hole No........... **OPERATING DATA** ....Mine

PROSPECT DRILL HOLE RECORD

Date Started:............ ....Date Finished:.................; Driller: . . ..

Helper:.....................

Total Depth:............Ft. Values?............ Followed Up?..........First Shipment.......

No. Machine Shifts:............ No. Man Shifts:.............. Total Labor Cost $....... .

Adv. per Mach. Shift:........... Adv. per Man Shift:.......... Labor Cost per Ft. $. ....

Average Cost per Man Shift: $..........

Remarks:...

....

....

....

....

....

....

....

....

....

....

Record Complete:..................19....

By ....

PROSPECT DRILL-HOLE RECORD FORMS USED AT THE CHIEF CONSOLIDATED.

COAL-MINING METHODS

Coal-mining Methods, with Especial Reference to Improved Methods and Higher Extraction

MEMBERS of the Institute comprising the Mining Methods Committee for coal and the suggested outlines for papers on coal-mining methods follow:

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Suggested Outline for Papers on Bituminous Coal Mining Methods*

Foreword

The Mining Methods Committee during the past year has spent much of its time developing the interest of members of the Institute in the work that comes under its direction. The response on the part of the membership has been most encouraging and in order to give a concrete form to the program which it hopes to develop, an "Outline for Papers on Mining Methods" has been drawn up and is herewith submitted for consideration. This outline has slowly taken shape until it now represents ideas contributed by a great many men, and it is hoped that it will be suggestive and helpful in the preparation of papers. This program can be used for the preparation of papers covering single large operations or, preferably, as the basis for a description of methods and the presentation of results obtained either in different districts or in different branches of the industry.

The Committee presents this outline for the consideration of the various Sections of the Institute with the request that they will endeavor to interest the members of the Sections in covering comprehensively the mining operations with which they are connected.

It is not the plan of the Committee to publish a paper on every mine, giving all the details outlined, but rather that an engineer familiar with a district prepare the *general information* and engineers of each important mine in the district compile the *specific information* of the mine with which they are connected. The general information and the specific information shall be sent to the Committee, and with this information the Committee will prepare a paper on the results obtained under the various mining methods in the different districts. The Committee requests that all papers shall be compiled under the direction of the chief engineer of the company and then submitted to the general manager before acceptance for publication or compilation and tabulation with other similar data. Where the study of this subject approaches the fields of other standing committees of the Institute, the Committee invites them to carry on the work for it or with it as they may desire.

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* Issued February, 1923.

The committee, realizing the immensity of its field, proposes to divide its activities among a number of special committees, as follows:

Advisory Committee
Classification of Mining Methods
Anthracite
Bituminous Coal
Copper
Iron
Lead and Zinc
Precious and Rare Metals
Non-metallic Ores
Sampling and Estimating of Ores

On account of the size of the copper-mining field to be covered, three committees are provided for: one with headquarters in Butte, attending to vein deposits, a second covering the Lake Superior district, the third with headquarters in Bisbee, covering the so-called porphyries and the other massive deposits. The membership of these various committees is here set forth, and it is requested that all members interested communicate with the chairmen of these committees or with the offices of their local sections.

An outline of this kind is necessarily comprehensive and it is not the idea that all of the topics brought out should be covered in detail, but the important points in each operation or district should receive special attention. It is earnestly hoped that the membership of the Institute will coöperate with the Mining Methods Committee to the end that the TRANSACTIONS of the Institute may constitute a real record of progress in the industry.

OUTLINE FOR PAPERS

To be followed as closely as practicable by the engineer in each district. Important features not covered by this outline should, of course, be brought out, giving proper weight, however, to the fact that the information desired is a full and detailed write-up of the principal mining methods obtaining in the district, with sufficient data and other information to bring out the reason for its adoption in preference to other operating methods. Paragraphs 1 to 6 inclusive, dealing with general information of a district, may be written by one engineer in each district. Data in the balance of the outline may be compiled by another engineer or engineers from specific information obtained relative to the principal mining methods used in the district.

All the information should be combined in one paper accompanied by detailed drawings and sketches necessary to show clearly all interesting details incident to exploration, development and operation of property.

1. NAME AND LOCATION OF MINING DISTRICT.
2. EARLY HISTORY, briefly told, to date.
   State whether ownerships are held in fee or lease or both.
3. GEOLOGY OF THE DISTRICT, briefly told, giving age and kind of rocks, physical structure of coal seams, amount and character of overburden.
4. DESCRIPTION, briefly told, OF GENERAL ITEMS THAT WILL INFLUENCE METHODS OF MINING, such as topography of district, climate, altitude, water and fuel supply, timber and other material supplies, character and efficiency of labor (native or alien, union or non-union) accessibility, methods of transportation, possible pumping problems, workings under the sea, swamps, or rivers.
5. EXPLORATION, SAMPLING AND ESTIMATING METHODS used to determine character of seams preliminary to mine development.
   (*a*) Exploration: churn or diamond drilling, prospect holes and vertical sections.
   (*b*) Estimating: cubic feet per ton, mining losses, remarks.
   (*c*) Accuracy of methods as indicated by exploitation, to be stated if possible in per cent. of production over or below estimate.
6. HISTORY OF PRINCIPAL MINING METHODS FORMERLY IN USE, briefly told, and reasons for changing to present principal methods of district, such as the following: influence of union labor or of lease conditions, if any obtain; influence of marketing of the product, etc.
7. FULL DISCUSSION OF DISTRICT'S PRINCIPAL MINING METHODS. In districts where both underground mines and stripping pits are in operation, show application of factors determining whether coal should be mined by underground or stripping methods. Under the following two subheadings, give fully the more important details of the methods in use at the particular mines described, following the outline as nearly as practicable.
   (*a*) Underground mines.
   (*b*) Stripping mines.

Underground Mines

1. Briefly describe the more important features of the coal seam and roof that affect the operation of the mining method, such as the dip and roof conditions, as affecting support required and influence of these features and of any gases generated on ventilation problem.
2. MINE OPENING: SHAFTS OR DRIFTS. Size, depth, construction (timber, steel, concrete), reasons for location.

Give sketches of shafts with dimensions; operating features determining type; bottom and top arrangements. Show advantage of permanent location and construction in large plants mining large areas.

3. UNDERGROUND DEVELOPMENT PLANS. This may be shown by sketch giving plan of development with relative dimensions; gage, grade, radius of curvature and weight of rails of track; method of handling water.

4. MINING.

(*a*) Pick or machine undercutting. Type (chain or puncher); if chain, breast or continuous cutters (shortwall or longwall). Motive power, kind and amount per machine. Depth of cut. Feet of face cut or tons cut per shift. Effect of machine mining on blasting and on character of product. Cost of undercutting per ton. Percentage mined with machines and picks, respectively. Are shearing machines used. Entry driving machines: kind, rate of progress, results.

(*b*) Drilling and blasting. Types of augers. Size and depth of holes drilled; method of tamping and firing holes; kind and grade of explosives. Give special note if method is used to produce lump coal. Is lump coal or slack preferred. Does this affect plan of operations.

(*c*) Room and pillars. Description of room work and method of drawing pillars. State this in such manner that the tons extracted per foot of working can be calculated. Furnish necessary sketches. Are room pillars extracted as soon as adjoining rooms are completed or left for later extraction.

State whether or not coal is picked and, if so, whether in the mine or above ground; describe any equipment used in this connection.

(*d*) Timbering. Describe kinds used in headings and rooms, preservatives used, method of delivery to working places. Is any timber recovered for re-use. Describe steel or concrete supports. Stowing of waste to support roof; describe methods and results if system is used.

(*e*) Loading machines and scrapers. Are any used; if so, describe method of working.

(*f*) Haulage. Weight of rails used on main headings, branch haulages, and in rooms. Room track laid by loaders or by company men. Gage of track. Average and maximum grades. Where necessary to grade main haulages, what is maximum grade allowed? Size of spikes for different rail weights. Spacing of ties. Character of main side tracks or partings, distance from shaft or drift mouth, practice of advancing.

Mine Cars. Design of car used; capacity; size of axle; type of bearings; average number of trips per day of each car; cost of maintenance of mine cars in tons handled; average life. Are cars placed at face by company or by miners. Average distance from parting or side track to nearest room; to farthest room.

Locomotives. Size of locomotives used for main haulage; average length of haul; number of mine cars hauled per locomotive per day. Is gathering done by mules or locomotives? If by mules, are they used single or in teams? How many cars per day are gathered per driver? Are cars taken from face and placed to face of working place singly?

Type and size of gathering locomotives; number of cars handled per day; general arrangement of underground side tracks; average and maximum length of haul by gathering locomotives.

Rope Haulage. Tail rope or endless rope; character of drive; size of rope; speed; number of cars in train; cost of haulage.

(*g*) Underground storage and dumping. Pockets or large shaft bottom or partings; dumping direct to skip or through pocket; use of rotary dumps, automatic measuring pockets, etc.

(*h*) Hoisting. Type and size of hoists, cages and skips, capacities and hoisting speeds; safety catches on cages; size and type of hoisting rope; special features of importance; kind of power; hoisting in balance or not; method of caging.

(*i*) Pumping. Type and capacity of pumps; power.

(*j*) Air compression. Type and size of air compressors; kind of power.

(*k*) Ventilation. Type, capacity and location of fans, types of fan drives used, quantities of air circulated and usual water gages. Are air currents split? Are booster fans used? Percentages of gas allowed in each split. Character of permanent stoppings, doors, and overcasts. Methods of humidification. Precautions against accumulation of dust, methods of limiting areas of dust explosions.

(*l*) Lights. Open lights or safety lamps; types of safety lamps.

(*m*) Telephone. Location, lines in lead cables on floor or exposed on hangers. System used for general communication underground; method of signaling in haulage and in shaft.

(*n*) Transmission of electricity. Trolley and feed lines and method of support; use of cut-out switches or section insulators; automatic circuit breakers, type, territory controlled. Size and type of bonds on main headings and branch haulage roads, method of transmission of electric power to remote sections; voltage and type of lines used if alternating current is used in the mine.

5. Records of Unit Production.
   (*a*) Total production of period reported on (specify long or short tons).
   (*b*) Contract or day labor; contracts based on car or ton unit.
   (*c*) Tons per man-hour and also man-hours per ton for miners in headings.
   (*d*) Tons per man-hour and also man-hours per ton for miners working in room and pillars.
   (*e*) Tons per man per hour and also man-hours per ton for miners working on slate work or rock work.

Note.—Possibly cubic yards would be a better unit to use in this case.

   (*f*) Tons per man-hour and also man-hours per ton for all miners. (Takes place of (*c*), (*d*) and (*e*), if these are not answered.)
   (*g*) Tons per man-hour and also man-hours per ton for all underground day labor.
   (*h*) Tons per man-hour and man-hours per ton in transportation system.
   (*i*) Tons per man-hour and also man-hours per ton for surface labor (all working force on surface exclusive of office).
   (*j*) Tons per man-hour and also man-hours per ton for total organization including office force.
   (*k*) Classification of labor, expressed in percentage of total.
   (*l*) Labor turnover. Who hires and discharges, employment manager, superintendent or individual bosses.

6. Records of Units of Supplies used per ton of coal produced.

Note.—For purposes of this discussion only major items of supplies such as explosives, timber, and power (fuel or electricity) have been itemized, and all other supplies taken together. Some large companies do not carry important items (for instance timber) in their supply accounts, but as a separate charge, but for purposes of comparison same should be so included.

   (*a*) Explosives, pounds per ton.
   (*b*) Timber, feet B.M., linear feet or cubic feet per ton.
   (*c*) Horsepower, hours or kilowatt-hours (total) per ton. Also divided into following subheadings: (1) Mining (electricity or compressed air) per ton; (2) haulage and hoisting per ton; (3) pumping per ton; (4) ventilation per ton; (5) miscellaneous and shops.
   (*d*) Cost of supplies used but not taken into account above, expressed in percentage of total cost of production.

7. Recovery, when possible, give percentage of coal that is actually mined: (1) in area mined out; (2) in whole district.

8. Safety and Welfare Work, briefly told.

Note.—All figures under 5 and 6 assume fully developed mine operating under normal conditions. Figures under 6 assume that production costs cease with dumping of coal into railroad cars.

Open-pit Mines

1. Describe the more important features in connection with the seam that led to choice of method and affect the operation of the pit; topography in vicinity of deposit; depth and nature of overburden; size, thickness, and dip of seam; safe operating slopes in various materials, alignment and grade of tracks; location of repair and supply shops; water facilities for operating equipment, capacity of supply; is water treated; location and capacities of, and grades to, tipple drainage, on top of coal, after coal is removed; other points that may have important bearing; furnish sketches showing typical plant with pit plans and location of necessary tracks, buildings and waste dumps serving same.
2. STRIPPING.
   (*a*) Height and width of cut. Width of berm used under various depths of cover or overburden, with character of same. Is overburden blasted, how?
   (*b*) Machinery and equipment, giving number, type, size, and capacity of each of the following used: (1) Hydraulic giants; (2) shovels, steam or electric, weight, power plant, length of boom, dipper stick, capacity of dipper; (3) drag lines; (4) locomotives; (5) cars (drop bottom, side or end dump or solid end), with rotary dump, running gears, brakes, couplings, etc.; (6) locomotive cranes; (7) track lifters; (8) dump spreaders; (9) tie tampers; (10) drills; (11) tracks, gage, weight required for various types and weights of equipment.
3. MINING COAL.
   (*a*) Drilling, blasting. Type of drill, arrangement of holes, method of firing holes, kind and grade of explosives. What steps, if any, are taken to produce coal of quality to compete with product of underground mines.
   (*b*) Machinery and equipment, giving number, type, size and capacity of each of the following used: (1) Shovels; (2) drag lines; (3) locomotives; (4) cars; (5) locomotive cranes; (6) track shifters; (7) tie tampers; (8) drills; (9) tracks, gage, weight required for various types and weights of equipment; (10) tipples, kind and method of dumping, screens and sizes made, picking, washing and preparation; capacity.
4. RECORDS OF UNIT PRODUCTION, stripping operations.
   (*a*) Period reported on.
   (*b*) Total cubic yards stripped.
   (*c*) Ratio of yardage stripped to tonnage made available.

(*d*) Cubic yards per man-hour and also man-hours per yard for stripping crews.

(*e*) Cubic yards per man-hour and also man-hours per yard for entire portion of organization charged to stripping.

(*f*) Average cubic yards per varying sizes and types of shovels per hour for full operating time; also based on actual results per hour during period operated.

(*g*) Length of haul from shovel to dump.

(*h*) Grade of track.

(*i*) Average number of cars per train.

(*j*) Percentage of total time required for shovelling stripping.

(*k*) Percentage of total time required for hauling stripping.

(*l*) Percentage of total time lost (moving shovels, breakdowns, etc.)

(*j*), (*k*) and (*l*) should equal 100 per cent. of operating time.

5. Records of Unit of Supplies Used per yard stripped.

(*a*) Explosives, pounds per cubic yard; give strength and character of explosives used.

(*b*) Fuel, pounds per cubic yard, or express in terms of horsepower-hours per cubic yard, if possible. Where electrical power is used, express in terms of kilowatt-hours per cubic yard.

(*c*) Cost of repairs and maintenance expressed in percentage of the total cost of stripping per yard.

(*d*) Cost of supplies not taken into consideration above, expressed in percentage of the total cost of stripping per cubic yard.

Note.—All figures under 4 and 5 should be made covering various kinds of material encountered in stripping, such as surface overburden, broken rock, and solid rock where same occurs in such abundance as seriously to affect the cost of the operation.

6. Records of Unit Production, mining operations.

(*a*) Total production per period reported on (specify long or short tons used).

(*b*) Length of haul from shovels to tipple.

(*c*) Grade of track.

(*d*) Average number cars per train.

(*e*) Tons per man-hour and also man-hours per ton for pit crews.

(*f*) Tons per man-hour and also man-hours per ton for entire portion of organization charged to production, including mine office.

(*g*) Average tons per varying sizes of shovel per hour for full operating time, and also based on actual results per hour during period operated.

(*h*) Percentage of total operating time required for shoveling coal.

(*i*) Percentage of total time required for hauling coal.

(j) Percentage of total time lost (moving shovel, breakdowns, etc.)
(k) Classification of labor expressed in percentages of total.
(l) Labor turnover.

7. RECORDS OF UNITS OF SUPPLIES USED PER TON OF COAL PRODUCED.
   (a) Explosives, pounds per ton; give strength and character of explosives.
   (b) Fuel, pounds per ton or express in terms of horsepower-hours per ton, if possible. Where electrical power is used, express in terms of kilowatt-hours per ton.
   (c) Cost of repairs and maintenance of equipment expressed in percentage of the total cost of production per ton.
   (d) Cost of supplies used but not taken into account above, expressed in percentage of the total cost of production per ton.
8. RECOVERY. When possible, give percentage of estimated coal that is actually mined.
9. SAFETY AND WELFARE WORK.
   Reclaiming stripped territory for agricultural use.
   Is the spoil pile leveled off, are trees or any other vegetation planted; if so, what varieties? Has any state bureau or commission taken any steps or given advice toward reclaiming stripped territory?

PREPARATION

Sizes made, cleaning, picking on screens on tables; dry cleaning machinery, washing, jigs, tables, other types of washers, flotation, settling tanks. Cost of preparation.

NOTE.—All figures under 4, 5, 6 and 7 assume fully developed mine operating under normal conditions. Figures should assume, further, that stripping costs cease when waste material is deposited on waste dumps and coal costs cease when coal is loaded into railroad cars.

It is assumed that all figures called for above cover a period when the operation is conducted under normal conditions.

Suggested Outline for Papers on Anthracite Mining

PREPARED BY R. V. NORRIS, CHAIRMAN OF ANTHRACITE COMMITTEE

THE object of an investigation into the ultimate recovery practicable from coal beds is to furnish data on which estimated of yield can be based, which estimates will be more logical and reliable than those based on broad generalizations of percentage of yield.

Analysis of Subject

In making such an investigation, a careful analysis of the whole subject is essential; a mere list of estimated yields, taken from the results from exhausted areas or properties, while of value, is not sufficient; what is needed is a study of the causes that effect ultimate yield.

Variables

The final ultimate yield from any property must be profoundly influenced by many variables, and a study indicating the influence of each would be of inestimable value; such are:

Thickness of bed.

Character of bed; solid or with rock partings.

Character of coal, indicated by per cent. sizes.

Depth below surface: Average, maximum, minimum.

Pitch of bed.

Character of roof and bottom.

Character of overlying strata.

Influence of mining in other beds, above and below.

Systems of mining.

Rate of mining—that is, age of openings, and duration of operation. Whether rapid exhaustion has any effect on ultimate yield.

Physical conditions, a wet, dry, or gaseous.

Study of Factors

In studying the effect of the different variables, efforts should be made to find results comparable except for a minimum number of factors. For instance, a comparison of the ultimate yield of several beds of various thicknesses, all at approximately similar depths, with similar pitches and character of coal, would go far toward indicating the effect of thickness. A comparison of beds otherwise similar, but of varying pitches, would indicate the effect of pitch. A comparison of a flat thick

bed of hard coal with a thin steep bed of soft coal, would probably be valueless.

Data Needed

The data collected should be as complete as possible, and in all practicable cases should include, in addition to physical variables, an estimate of the original coal contents, the recovery in first mining, and the further recovery from second mining, and from later removal, the rate of mining, the amount lost, used for fuel, and the amount in banks. All bank recovery should be considered and added to the recovery from direct mining.

The methods of mining, size and arrangement of openings, and methods of second mining should be known, also the handling of top or bottom coal. The amount and effect of flushing if any, and the quantity and kind of materials available for this. The effect of mining and of removal of pillars in overlying or underlying beds should be noted.

It is important that the methods used in determination of the quantities of coal originally in the ground and recovered by mining, separated if possible into its various stages, should be given in detail.

Results

The results from as wide a study as is possible should be tabulated in comparative tables, first making separate tables for the effect of each variable, then as far as practicable combining these. Also plot curves of results as far as practicable.

It may be possible to find an empirical formula that will give effect to the principal variables and indicate the probable ultimate recovery under known or assumed conditions.

Losses in Mining Coal

In its reports to the U. S. Coal Commission in 1923, the U. S. Bureau of Mines stated that the average extraction from the anthracite mines was 61 per cent. and from ten states that produced 90 per cent. of the bituminous coal of this country the average extraction was 65 per cent. The range of extraction was from 49 to 69 per cent. for anthracite and 55 to 92 per cent. for bituminous coal. From actual observation it was estimated that the avoidable losses averaged 20 per cent., therefore the extraction should be increased by that amount. The reports further stated that there were signs that improved mining methods indicated a higher extraction in the future. The recent literature on coal mining bears on this phase of the problem. The large losses of coal, in other words the low percentage of extraction, have been the subject of consid-

erable discussion and investigation. The State of Illinois has been prominent in this work, and the late H. H. Stoek made a critical study of the factors affecting the percentage therein [*Trans.* (1923), **68**, 305].

Acknowledgement is here made to Mr. J. W. Paul, chief of coal-mining investigations of the U. S. Bureau of Mines, for his review of these papers, part of which is included in the summaries following:

Anthracite Mining: Summary

Losses

In one of the papers on anthracite mining, Otto discusses the factors that affect extraction. In the mining of more than 30,000,000 tons from sixteen coal beds ranging from 3 to 30 ft. in thickness, the average removal (that is, coal taken from the mine) was from 72.3 to 57.5 per cent. The thinner the bed the higher was the removal; for example, from a 4-ft. bed the removal was 75.1 per cent. and from a 30-ft. bed it was only 57.5 per cent.

The geologic factors affecting recovery are: thickness and character of bed, character of coal, depth below surface, dip of bed, character of roof and bottom, character of overlying strata, influence on mining of overlying and underlying beds, surface wash and streams. The wetness and gaseous condition of a bed are also factors of importance.

The underground losses in mining and preparing anthracite are: coal left in pillars for support of shafts, slopes, tunnels, etc.; coal in barrier pillars; coal left for surface support of streams, towns, cities, highways, and railroads; coal lost in robbing because it cannot be removed with safety; coal lost through breaking of roof; coal lost in gob; coal lost through fires, squeezes, and floods; coal lost through misuse of explosives; and coal lost in transportation. The reasons for these losses are discussed and how some of them can be and are being lowered.

This paper calls attention to the lack of engineering study in the early development of the deposits which is now responsible for losses that cannot be corrected except in virgin ground, and of this there is little remaining. No attempt is made to discuss the details of any of the mining methods, but the data presented should be a stimulus to the anthracite operators to devise measures whereby methods may be improved to insure a greater recovery of the original deposit.

Removal of Pillars

When mining an anthracite vein with a pitch of 65° to 70°, Miller states that the breast-and-pillar method has produced good results as far as mining the breast was concerned, but the removal of the pillar was difficult as to full extraction and expensive as regards costs. Three methods have been employed in removing pillars after breasts were

driven, and the pillar-chute system was most commonly used. This is expensive and much coal is broken while running down the chute. In a new method, slant chutes are driven. Breasts are opened off these chutes, then auxiliary chutes, and then skips. This method allows of complete removal of the vertical pillar between the main slant chutes without any of the objectionable features of the other methods. The extraction is 68 per cent., with 32 per cent. remaining in pillars and stumps. By this method, the output from a given development was more than doubled at half the cost of other methods of pillar extraction, with an advantage of an increase of 24.3 per cent. in prepared coal. The method appears to offer the miner greater safety.

Reducing Maintenance Charges

Complete extraction in panel by first mining in steep-pitching anthracite beds as described appears to have both safety and economic advantages. By the method used, the ventilation of the face of the breast should be effective; freedom from falls of rock is dependent on the top coal of 11 ft. remaining in place until the breast is drawn. No statement is made concerning the action of the timber when the top coal is shot down, whether it tends to block the battery when it slides with the coal. The method offers economy in the upkeep of the gangway, improves the prepared sizes, and recovers all the coal according to Ashmead.

Mining Bituminous Coal: Summary

Alabama Practice

When mining up to 20,000,000 tons of coal a year, Alabama operators face many problems. Coal is won from thirty workable beds, which range in thickness from 2 to 13 ft. and pitch from 0 to 90°; the character of the beds varies considerably, and about 60 per cent. of the coal mined must be washed and cleaned. Fies describes the methods in vogue for mining thin, medium, and thick flat beds, medium and steep-pitching beds, and strip mining. The average removal of coal in Alabama is 68 per cent. but it is steadily increasing in a number of mines, especially in those of large companies, through improved mining systems.

The features of most interest in this paper relate to methods of mining that admit of concentration of work, the use of mechanical means for loading the coal into the mine car, and system of roof supports for longwall mining, all of which appear to contribute to maximum recovery of the coal and at a cost much under the regular room-and-pillar method which results in loss of coal.

Large Producing Unit in Illinois

The New Orient, a comparatively new mine in Illinois, has been equipped and is being developed for an ultimate daily output of 12,000

tons—that is, in 8 hr. It has already produced 8200 tons a day, according to Harrington. A horizontal coal bed 9 to 12 ft. thick is being worked and the mine has many features not common practice in shaft mining.

Certain economics are presented in this development, the principal ones being rapid transit, the use of a 6½-ton capacity mine car and skip hoist. To obtain the economy offered the mine must run to capacity with great regularity. Without a market that will absorb the daily capacity of the mine and its mechanical equipment it would appear that the capital investment may be questionable.

The area to be worked over is 10 sq. mi. or 6400 acres. Under the maximum daily capacity of 12,000 tons, there will be exhausted a mine area of 1.5 acres per day; and by working 300 days per year the entire area will be exhausted in 14 years, with a loss of 2240 acres of coal left in the form of pillars, representing 22,400,000 tons of coal. At 10 cents per ton this would total $2,240,000, or sufficient to purchase all of the surface of the 6400 acres at $350 per acre.

Washington Practice

Many difficulties are encountered in mining coal in the Washington, according to Ash, who describes underground methods that have been found successful as well as those that have not. In Pierce County, the beds dip from 65° to 70°. One company with a 4 to 4½-ft. bed dipping 38° works it by the longwall method, and has reduced the cost so that a profit is made, whereas by breast-and-pillar and chute-and-pillar methods a loss resulted. Another company has a double, or twin, bed that pitches from 45° to 70°, and has 66 in. of coal in the top bench, 64 in. in the lower bench, and 91 in. of parting, mostly shale. Here the chute-and-pillar method is used and the removal of coal is good. The lower bench is worked advancing and the top bench retreating. In a third property, the five beds have an average dip of 40° and the retreating system is used.

The methods described cover steep-pitching beds in the bituminous and subbituminous coals of the Eocene period, which carry shale laminations in different parts of the section of the coal bed.

It is of interest to note the progress of machine mining in the steep-pitching beds, the post-puncher type being in use.

West Virginian Practice

In the Pocahontas field, according to O'Toole, the panel system was first adopted for the development of the largest property, but it was found to be defective, particularly as to expense and loss of coal in barrier pillars. Now, the general plan is one of room-and-pillar continuous advancing and retreating. No standing pillars, with resultant open work to be ventilated, are left for later difficult and expensive extraction.

The beds dip from 3 to 8 per cent. and are mostly mined through drifts. The average production per man-hour for all miners is 1.91 tons.

A systematic organization for safety has been introduced for all departments, based on a study of the probable hazards, and premiums are paid monthly to underground foremen for good records in prevention of accidents. The welfare work done by the operating company appears to be reflected in a small labor turnover.

The regularity of tonnage production and the economies realized appear to have been accomplished by the application of the engineering profession in a study of the many details of the problems that enter into mine development and operation.

Ultimate Recovery from Anthracite Beds

By Henry H. Otto, Lansford, Pa.

(New York Meeting, February, 1925)

The anthracite industry can be divided into two parts—the underground, or mining, and the outside, or preparation or manufacture. To understand recoveries in the two branches, some of the history of the industry and the changes that have taken place in the past century should be considered.

It is a long step from 1807, when 55 tons were produced in the Wyoming field, to 1917, the peak year, when 80,841,223 tons were shipped to market from all the fields. In 1820, the Lehigh Coal Co., the predecessor of the Lehigh Coal & Navigation Co., shipped 365 tons in arks down the canal of the Lehigh Navigation Co. The Schuylkill field was not opened until 1822, when 1480 tons were shipped. The Wyoming field really became a producer in 1829, when 7000 tons were mined and shipped.

It is well to note, at this time, the passing of the Coleraine colliery after a life of 88 years; in collieries of this age will be found the romance of the region, as well as the history of the changes to date.

Mining Methods

In the Lehigh region, the first coal came from the old quarry mines at Summit Hill; a little later the open-cut workings, or quarries, east of Jeansville, and near Beaver Meadow were opened. In the Wyoming field, it was necessary to resort to underground mining almost from the very first. Ashmead[1] has described the changes in outside preparation methods, all tending to increase the percentage of coal shipped to market. A description from the days when the lump coal was raked and screened out in the quarries or mines, to the modern breaker of today, shipping as high as 6000 tons of at least eight sizes of coal in an 8-hr. day is a very interesting story. Each size is a step toward increasing the yield from the property.

With the belief that the supply was inexhaustible, the early mining was conducted with the thought of getting the maximum amount of coal on first mining. The superintendent was his own foreman and engineer.

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[1] Dever C. Ashmead: Advances in the Preparation of Anthracite. *Trans.* (1921) **66**, 422.

As the mining became deeper and the production larger, however, it was necessary to give the superintendent assistance. Today, the mining engineer is almost entirely responsible for the proper laying out of the mines, the aim being to obtain the maximum economic recovery. In recent years, some companies have also employed men, known as robbing inspectors, to inspect the robbing area between general surveys or postings, and protect the operator and the land owner from waste by careless mining; frequently the inspector must show the miner how to pull back pillars without endangering the life of the latter.

The three classes of mining in the anthracite field are: stripping, which yields the maximum recovery; flat mining, where a partial separation of coal and refuse is made inside; and steep pitch mining, where the coal and refuse are so mixed that both must be shipped together to the breaker for a proper separation. The last naturally yields the smallest maximum recovery.

In the early days, no robbing of any consequence was done in either the flat or the light pitch mining; large areas were mined and the pillars were reduced to a minimum for the required support. This mining was most skilfully done, as later attempts to rob these areas revealed, and as a result considerable coal can now be recovered only at a high cost and in periods when the market is exceptionally good. The Hillman, Baltimore, and Red Ash seams in the northern field, and the Mammoth and Buck Mountain beds in the middle and southern fields were treated in this manner. Conditions of this type make the problems of the mining engineer and superintendent very difficult.

Disregarding surface support requirements, such as support for towns, rivers, and streams, the geological conditions of the Wyoming, middle and southern fields make it impossible to coördinate them into one general group in determining the various losses.

Modern Mining Methods

The maximum recovery undoubtedly is obtained from strippings. The earliest hand strippings in which only the clay was removed, helped to increase the yield from some properties; but with the increased size of stripping equipment it is possible today to strip entire basins, which only 15 or 20 years ago it would have been impossible to do, and thereby obtain a maximum recovery.

In the flat territory, for the thicker seams the room-and-pillar method is generally used. An increasing number of the thin seams is being mined by modified longwall methods. In the lighter pitch territory, a high percentage of the marketed coal is recovered by stripping. The underground mining is conducted as a breast-and-pillar method, the centers of breasts as a rule being 50 or 60 ft. apart. In the southern heavy pitching

fields, some coal is recovered from strippings. The inside conditions require a breast-and-pillar or chute-and-pillar method, and the mining on the upper levels must be well out of the way before work is started on a lower level. In this territory, the thin seam gangway and rock gangways have been developed to replace the regular coal gangways in the seam to be mined. Where a thin seam is within 20 to 30 ft. of the Mammoth bed and has a good roof, the gangway is driven in the thin seam, chutes are driven up the pitch, and tappings are made into the seam to be mined. If the thin seam is too close to the Mammoth or rock conditions do not permit, the gangways and chutes are driven in the rock. This method has already been described;[2] it permits a much higher extraction than is obtained when the gangways and chutes are driven in the Mammoth or Primrose seams.

Factors Affecting Recovery

The geologic factors affecting recovery are: Thickness of bed, character of bed, character of coal, depth below surface, dip of bed, character of roof and bottom, character of overlying strata, influence on mining of overlying and underlying beds, surface wash and streams. These are supplemented by the physical conditions, such as whether or not a seam is wet or gaseous. The main factors determine, to a large extent, the losses in first mining and robbing, and will be discussed later.

The inside losses in mining and preparing anthracite are as follows: Coal in pillars left for support of shafts, slopes, tunnels, etc.; coal in barrier pillars; coal left for surface support of streams, towns, cities, highways, railroads; coal lost in robbing because it cannot be removed with safety; coal lost through breaking of roof; coal lost in gob; coal lost through fires, squeezes, and floods; coal lost through use of explosives; coal lost in transportation.

The outside losses are: Transportation loss, loss of coal in refuse, loss of coal in silt, loss of coal used as boiler fuel.

The loss of coal for support of shafts, tunnels, slopes, etc., will depend largely on the location of the shaft, or tunnel to be supported, and will vary in each colliery and with the depth.

The loss in barrier pillars is necessary for safety and economic reasons; when collieries are in the last stages of exhaustion, the necessity of good barriers is felt by the adjoining collieries.

It is necessary to leave large quantities of coal in place to support highways, railroads, towns, cities, streams, and rivers. Some leases specify the percentages of coal to be left for surface support, others leave the amount up to the operator. Culm and rock filling are used to reduce

[2] W. G. Whildin: Steep Pitch Mining of Thick Coal Veins. *Trans.* (1914) **50,** 698.

the percentage of coal to be left for surface support. A free-falling slate roof is a large factor in obtaining a high extraction where the surface must be supported. A number of shafts have been permanently lost because of inrushes of sand and water from the Susquehanna River or from the Buried Valley of the Susquehanna; these losses are reduced, in part, by careful diamond-drill provings for rock-cover limits. In other regions, collieries have been temporarily lost as a result of small streams breaking into the mines.

In robbing, many small portions of pillars are lost because they cannot be recovered with safety; at times, large areas are lost because the robbing is hurried and improperly conducted.

In first mining, in both flat and pitch breasts, coal is lost on account of a soft falling roof. The loss in pitch mining is considerably higher than that in the flat territory, and is reduced by driving narrow breasts or chutes, which are usually driven at a higher cost than full width breasts.

Coal lost in the gob in the Wyoming field depends on the thickness and section of the seam, the care of the miner in the use of explosives, and the separation of the coal from the refuse. A test made when cleaning up 60 ft. of breast in a bed with the following section,

	Feet	Feet
Bone		0.4
Coal	1.1	
Bone		1.2
Coal	2.5	
Total	3.6	1.6
Bed	5.2	

revealed a loss of 8 per cent. of the original coal content. All of the gob was loaded into cars, and the floor was swept, and the material was sent outside, where it was carefully separated and the coal weighed. In the smaller dirty seams, the gob loss is high. Gob in the southern anthracite field means a mixture of coal and rock, caused by the caving of the roof over a breast. The gob loss is naturally high.

The losses through squeezes, floods, and fires are very important, and in a measure are within the control of the operator. There have been many squeezes, particularly in the Wyoming region. Many were caused, not by careless engineering and mining, but by unknown geological conditions, which became known after the mining had progressed. If large reserve pillars had been left and no mining done in them, except gangways and airways, squeezes could have been localized, thereby reducing the total loss. The same is true of the lighter pitching areas. In the heavier pitching areas, safety requirements necessitate the practical exhaustion of a level before going lower, hence large squeezes in unexhausted areas are prevented.

The fire losses, however, are greater in the pitching territory than in the flat territory. The Carbondale fire is about the only large fire in the flat territory; the Red Ash fire is on the lighter pitches. The fires in the Panther Creek valley and at the Sioux colliery near Mt. Carmel are typical examples of tremendous losses as a result of mine fires. The causes of some of the fires are unknown; they are not the results of carelessness with lamps, etc., but may have been the result of improper firing by the miner; in the softer measures they may be caused by the oxidation of the pyrites, although the latter has not been definitely proved. The losses by fires, floods, and squeezes are among the big mining risks. No percentage can be assigned to them but the history of the region shows that they are an exceedingly important factor in the mining loss.

The use of modern dynamites, instead of black powder, results in a greater degradation in the blasting of the coal, causing a higher percentage of silt. Carelessness in drilling and firing on the part of the miner has caused additional loss by coal being shot into the gob.

Poor mine cars and carelessness in handling modern haulage equipment causes considerable coal to be spilled along the roadways. The transportation loss, as a result of leaky mine cars, is high in collieries producing less than 55 per cent. prepared sizes; this loss is being reduced by the use of rotary dumps and closed-end cars.

The inside losses are being reduced by mining beds in proper sequence, by columnizing workings, and by careful supervision and frequent inspections. Back filling of breasts in the flat territory, by silting or otherwise, has made considerable pillar coal available for market.

The outside losses are much more easily controlled than the inside losses. The outside transportation loss is smaller than the inside transportation loss, because roads outside can be more easily kept clean than those inside. The refuse loss depends on the breaker equipment being of sufficient capacity to do the work properly, on proper maintenance of the same and on making necessary breaker changes to increase the recovery. Coal lost in refuse will run from less than 1 per cent. upward.

The silt loss depends, first, on the softness or friability of the coal and, second, on the breaker equipment. Low silt losses prevail in the Wyoming field, and high losses in the main southern field. Silt should not be treated as a waste product; wherever possible, it should be stored for future use in manufacturing briquettes or possible use as a pulverized fuel. Griffen[3] gives a summary of the determinations of solids in silt as follows:

	Wyoming Field, Per Cent.	Lehigh Field, Per Cent.	Schuylkill Field, Per Cent.
Total solids	7.2	22.4	22.3
Recoverable coal, with 15 per cent. ash	3.5	9.5	8.8

[3] John Griffen: Slush Problem in Anthracite Preparation. *Trans.* (1921) **66**, 514.

The percentages given express a ratio of solids to shipments. Some actual tests show solids in slush at individual collieries as follows:

	Per Cent. of Tonnage Shipped
A, northern field	8
B, southern field	23
C, southern field	22
D, southern field	28
E, middle field	19.9
F, middle field	17.0

In the Southern field collieries about one-third of the silt was coal, and in the Middle field collieries about one-half was coal, which could have been recovered on concentrating tables.

Boiler fuel is, roughly, 10 per cent. of all the coal mined in the anthracite region; it is generally regarded as a loss, although the tendency is to charge it against the cost of steam at the prevailing price for the particular kind of fuel used. The highest grade of fuel is used by steam shovels, locomotives, and blacksmiths. The fuel loss is being reduced by the use of purchased electric power or modernizing the steam-generating equipment, supplemented by the installation of the best hoists, compressors, and pumps, which are the largest users of steam. At some collieries, where a large amount of water is pumped in comparison with the coal shipped, the boiler fuel runs as high as 22 per cent.; at one colliery it ran as high as 35 per cent. in the closing years of the operation.

Discussion of Factors Affecting Mining Extraction

After carefully examining considerable data giving tests for limited areas, it was found impossible to plot the data into curves that would

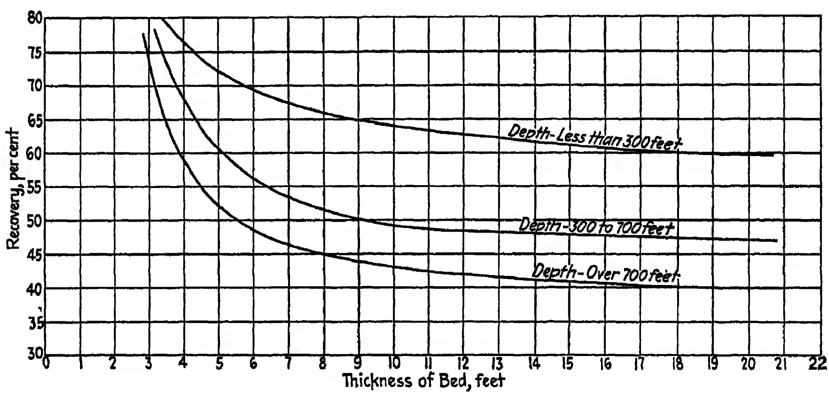

Fig. 1.—Trend of variation in extraction with variation in depth and thickness of bed.

show the effect of pitch, thickness, depth, etc., on account of the big variations in the conditions in a single seam, and also largely because

good records have not been kept of the shipments by beds. However, several curves showing the general trend of the variation in recovery, with change of depth and bed thickness, have been plotted; the curves (Fig. 1) show that the trend is toward a decrease in removal with increase of depth and bed thickness. It is to be emphasized that these curves are of no actual value, except to show the general tendency. If more data were available, empirical formulas could be derived to show this tendency with a degree of accuracy not expressed by these curves. The effect of dip might be considered in such formulas and perhaps other factors, such as the character of the coal. It is necessary to have considerable data of a reliable character before deriving such formulas.

The average extraction from different areas in all the anthracite fields for beds of given thickness, regardless of dip, depth or other factors was as follows: the total production from the combined areas was over 30,000,000 tons.

Extraction in Mining Anthracite Fields

Bed Thickness, Feet	Tonnage	Average Removal, Per Cent.
3	745,159	72.3
4	2,382,827	75.1
5	4,550,819	71.0
6	4,093,994	71.5
7	3,190,762	70.6
8	1,859,420	58.6
9	2,567,181	72.8
10	2,203,678	72.1
11	2,515,154	60.8
13	2,332,548	68.9
15	291,800	73.4
17	695,870	65.6
19	287,495	72.0
22	498,600	67.7
27	255,579	62.5
30	2,307,455	57.5

If more data had been received for seams averaging from 15 to 19 ft. thick, the percentages for these seams would, in all probability, have been lower.

Many factors affect recovery. The information furnished us represents so many conditions in the different fields, that we can plot the bed thickness against the recovery to determine the most probable curve to represent the average effect of thickness on extraction (see *I*, Fig. 2).

The formula, $R = 74.24 - 0.45t - 0.00055t^2$, represents the most probable value of the recovery R, in per cent., for a bed thickness of t feet. If better than average conditions prevail, the percentage obtained by using the formula should be increased; if the opposite were true, the percentage should be reduced. For seams 50 ft. thick, dropping the term $0.00055t^2$ will increase the probable recovery only 1.4 per cent. As the

probable error of the formula is ±3.4 per cent., this term can be dropped for general forecasting. The formula is subject to further reduction for outside losses, such as culm, breaker and fuel losses.

For the northern field, a reasonable amount of data has been submitted to show, in a measure, the effect of depth on recovery for all veins, regardless of thickness, dip and other factors. The data, *II*,

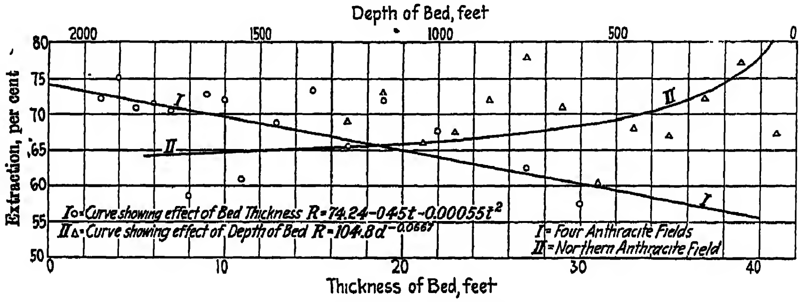

FIG. 2.—EFFECT OF DEPTH AND THICKNESS ON EXTRACTION.

Fig. 2, was obtained for depths up to 1300 ft., and represents the average of a number of limited areas. Most of the tonnage was won from beds not more than 500 ft. deep. This curve shows the trend of the effect of depth.

Table 1 shows that in the Wyoming, or northern field, the Baltimore and its various splits had more squeezes than any other seam, and that the Red Ash seam was next. Beginning at the Lackawanna-Luzerne County line, the Red Ash was subject to a squeeze in every colliery mining in the main basin to and beyond the junction of the Lackawanna with the Susquehanna River. On the west side of the river, the squeezes predominated in the Ross seam; on the east side, the squeezes were largely in the Baltimore and Hillman beds. They seemed to have been caused principally by trying to win too high a percentage of coal on first mining in the main seam, and then mining another seam or split contiguous to the main seam. As a result of poor columnization of the workings, the squeezes were started where the percentage of extraction in the main seam did not appear to be excessive.

We were able to secure some complete data giving the coal marketed from five collieries; unfortunately, the tonnage in the early years was not kept by seams. Estimates have been carefully worked up and the overall percentages of the original coal marketed from the minable area are submitted. Fault and barren areas have not been included with the minable areas.

TABLE 1.—*Tabulation of Squeezes*

Colliery Number	Vein	Thickness, Feet	Pitch, Degrees	Area, Acres	Per Cent. Mined	Date of Squeeze	Depth Below Surface, Feet	Cause of Squeeze	Remarks
1	Red Ash	7.0	8	90	54		400		
2	Red Ash	12	8	129	62		250		
3	Red Ash	6.0	10	180	66		335		
4	Red Ash	5.7	5–10	9	51	1897	1000–1500		24 ft. breasts, 30 ft. pillars, shelly coal, gas appeared in great quantities.
5	Red Ash, top	10.0	5–10	29	52	1897	1000–1500		
6	Red Ash	11.2	5	86	58		520		
7	Red Ash	6.7		100	48		1000–1500		
8	Red Ash	6.5	5–8	210	62.5	1896	433	Irregular mining, squeeze precipitated by mining fifth vein.	Interval to fifth vein 10 to 30 ft., fifth vein 3 to 4 ft. thick.
9	Red Ash	16.0	5–15	85	50		350		
10	Red Ash	21.0	10–15	40		1894		Irregular pillars, coal friable; pillars chipped off.	
11	Red Ash	16.0	5–15			1890			Broken to surface, admitted water from Susquehanna.
12	Red Ash	21.0	5–15			1897		Irregular mining.	
13	Ross	6.2		83	46		500–1000		
14	Ross	18.0		94	43		500–1000		
15	Ross	7.0	5–15	49	69		200		
16	Ross	4.5	8	42	70		280		
17	Ross	4.0	8	19	60		310		
18	Ross	6.8	5–15	162	61		160		
19	Ross, top	3.5	8	42	70		240		
20	Ross, top	3.3	8	19	58		290		
21	Baltimore, Cooper	6.5	8	50	50	1919	415–525	Irregular mining, no columnization.	24 ft. chambers.
	Bennet	6.0							
22	Baltimore, Cooper	6–9			48.7				
	Bennet	6.0	0–15		51.1	1916	730–790	Poor columnization, chamber too wide.	Reserve pillar every 11 or 12 breasts.
23	Baltimore, Bennet	10.0	5–10	26	54		500–1000		
24	Baltimore, Pittston	11.0	8	120	61.5		400		
25	Baltimore, Pittston	11.75	8	131	62.6		260		
26	Baltimore	14.0		64	40.6		900		

27	Baltimore	16.0	15			1889	700		Pillars large, breasts narrow; gas appeared in great quantities before collapse; squeeze started in Bennet and affected Cooper, Five Foot, and Hillman.
28	Baltimore, Cooper Baltimore, Bennet		10–15	100		1899			
29	Baltimore, Top Baltimore, Bottom		5–10			1884		Small interval, lack of columnization; abandoned Hillman full of water.	
30	Baltimore		15–17	10		1874		Irregular mining, fire, no reserve pillars to break squeeze.	
31	Baltimore, Mammoth	30.0	25			1877		Irregular mining, extensive robbing; no reserve or gangway pillars.	Surface dropped 11 ft.
32	Top Baltimore	6.0	20–30			1890			Interval to bottom split 15 ft., bottom split unaffected.
33	Top Baltimore, Cooper		10	4½		1890			Cogs and slush prevented squeeze from spreading.
34	Baltimore, Top Mammoth Bottom	6.4	15–35	16		1918	200–400	Too many pillars skipped.	24 ft. breasts, 55 to 60 ft. centers; interval 10 ft. of soft slate and fire clay.
35	Hillman	8.2 9.8	8	30	50	1919	415–525		Too many roads through reserve pillars, 24 ft. chambers, 50 ft. centers.
36	Hillman	10–12	10–15	20	50	1918	340–570	Pillars too narrow; extraction in first mining too high; columnization poor.	24 ft. chambers, 50 ft. centers.
37	Hillman, Lance	6.0			48.6	1916	730–790	Poor columnization; chambers too wide.	
38	Hillman	7.5		88	53.8		550		Precipitated by Bennet Cooper squeeze.
39	Hillman, Lance	6.1	10	18	82.9		500		
40	Hillman	7.0	5–15	85	50		350		
41	Hillman		15			1899	400	Due to squeeze in Baltimore below.	
42	Kidney	5.5	15–40	20		1879		Cover limit, too shallow.	
43	Diamond and rock (Contiguous)					1878		No columnization.	

TABLE 2.—*Data Relative to Interval, Thickness and Character of Veins and Method of Mining Employed at Colliery A*

Vein No.	Interval, Feet	Thickness		Character of Bed[c]	Character of Roof	Character of Bottom	Overlying Strata[c]	Influence of Mining	
		Feet	Inches					Above	Below
2[a]	140	2	6	C. 1′ 6″ S.C. 1′ 0″	Very hard	Slate	Conglomerate	None	None
3	220	2	6	C. 2′ 0″ S.C. 0′ 6″	Very hard	Slate	Conglomerate	None	None
4	80	3	6	C. 2′ 0″ S. 0′ 6″ C. 1′ 0″	Hard	Slate	Sandstone	None	None
5	30	3	9	C. 1′ 9″ S. 0′ 8″ C. 1′ 4″	Hard	Slate	S. Slate	None	None
6	70	2	6	C. 2′ 6″	Hard	Slate	S. Slate	None	Falls into No. 5 vein
8	10–30	7	8	B. 0′ 8″ C. 2′ 0″ B. 1′ 0″ C. 4′ 0″	Hard	Slate	Conglomerate	None	None
9	30–40	7	7	C. 0′ 10″ S. 0′ 3″ C. 0′ 6″ S. 1′ 0″ C. 5′ 0″	Friable Slate	Slate	Slate	None	Strata breaks to No. 8 vein
9½	80	7	4	C. 0′ 6″ S. 0′ 5″ C. 1′ 10″ S. 0′ 3″ C. 4′ 4′	Friable Slate	Slate	Slate	None	Strata breaks to No. 9 vein
9¾	40	4	11	C. 3′ 6″ S. 0′ 5″ C. 1′ 0″	Clod	Slate	Clod Sandstone	None	None
10	50	5	3	C. 5′ 0″ S.C. 0′ 3″	Clod	Slate	Clod Sandstone	None	Strata breaks to No. 9¾ vein
10½	40	2	6	C. 2′ 6″	Slaty	Slate	Slate and sandstone	None	None
11[b]		5	8	C. 4′ 0″ S. 0′ 8″ C. 1′ 0″	Slaty	Slate	Slate and sandstone	None	None

[a] Bottom vein worked.
[b] Top vein worked.
[c] Designation: C, coal; SC, shelly coal; B, bone or bony coal; S, slate; S. Slate, sand slate.

TABLE 3.—*Data Relative to Interval and Character of Beds at Colliery B*

Vein	Minimum Dip, Degrees	Maximum Dip, Degrees	Minimum Interval, Feet	Maximum Interval, Feet	Thickness of Vein, Feet	Thickness of Coal, Feet	Remarks
Orchard.......	40	87	130	150	6.0	5.5	Fair vein, full of slips, breaks easily
Primrose......	40	70	50	100	5.0	3.5	Poor quality, very faulty; low per cent. of prepared sizes.
Holmes.......	40	65	100	130	8.0	8.0	Good vein, hard coal in spots, only worked where good
Four foot......	37	55	80	100	5.0	4.0	Good coal in limited areas where worked; bottom bench dirty.
Top split......	35	55	60	80	5.5	4.0	Vein full of rolls and dirty.
Middle split...	29	64	30	70	3.0	3.0	Good hard coal.
Bottom split...	34	62	100	110	10.0	9.6	North basin, good hard coal, balance soft but fair.
Skidmore......	40	45	60	70	3.5	3.0	Poor vein, not exploited.
Seven foot.....	40	45			3.5	2.8	Fair coal where developed.

TABLE 4.—*Character of Beds, Intervals and Methods at Colliery C*

Vein	Section	Interval, Feet	Top Rock	Bottom Rock	Remarks
Primrose........	Coal 3.7 ft. Slate 0.7 ft. Coal 3.3 ft.		Hard sandstone	Slate	Coal hard
		60			
Mammoth.......	Coal 15.0 ft. Poor Man 3.0 ft. Coal 9.5 ft. Slate 1.0 ft. Coal 5.5 ft.		Hard sandstone	Hard slate	Coal hard
		90			
Wharton........	Coal 5.5 ft. Bone 1.2 ft. Coal 3.5 ft.		Hard sandstone	Hard slate	Coal hard.
		140			
Gamma.........	Coal 5.0 ft. Bone 1.3 ft.		Hard sandstone	Hard slate	Thin, faulted, and unworkable over large areas.
		50			
Buck Mountain..	Coal 2.4 ft. Slate 0.2 ft. Bone 0.3 ft. Coal 0.6 ft. Slate 0.2 ft. Coal 0.5 ft.		Hard sandstone	Hard slate	Thin, faulted, and unworkable over large areas.
	Beds dry and non gaseous, No slushing or rock filling.				

First mining, room-and-pillar.

Robbing { pillars skipped / pillars split } beds robbed in descending order

TABLE 5.—*Character of Beds, Intervals, etc., at Colliery D*

Vein	Thickness, Feet	Depth, Feet	Per Cent. Removed	Top Rock	Bottom Rock
Checker	5.8	136	60	Hard fireclay	Hard sandstone
Pittston	10.0	210	60	Slate	Hard sandstone
Top split	4.3				
Bottom split	2.8	155	60		
Marcy					
Top split	2.5	230	65	Hard slate	Hard slate
Bottom split	3.6	250	65	Soft sandstone	Soft fireclay
Red ash	6.7	467	60	Soft	Hard

TABLE 6.—*Character of Beds of Colliery E*

Vein	Thickness, Feet	Top Rock	Bottom Rock	Remarks
Bowkley	5.0	.....	.....	Small area
Hillman	11.0	.....	.....	Small area
Orchard	4.5	Good	Good	Dip 8° to 12°
Five foot	5.0	Poor	Hard	Dip 8° to 12°
Cooper	7	Soft	.....	Dip 8° to 12°
Bennet	12	.....	Soft	Dip 8° to 12°
Ross	7	Hard	Hard	Dip 8° to 12°
Red ash	20	Soft	Hard	Dip 8° to 12°

Colliery	Coal Area Acres	Pitch	Number of Beds Mined	Maximum Depth, Feet	Shipments, Long Tons	Per Cent.
A	1143	Heavy	12	900	7,495,752	43.3
B	370	Heavy	7	700	2,320,200	42.7
C	346	Medium	5	500	11,418,843	61.8
D	307	Slight	6	467	5,047,994	51.0
E	155	Slight	8	810	5,068,118	45.5
					31,350,907	

Colliery A is in the western middle field and can be considered a typical colliery. There are no towns over the area, merely roads, streams and railroads. Thirteen seams of coal lie in two main basins; the southern basin is deep and the dips heavy; the northern basin is shallow and not seriously folded; the intervals between the beds are not large. A few strike faults run east and west across the property. About 43.3 per cent. of the coal in the ground from the areas considered was shipped to market, 26 per cent. of the original coal was recovered on first mining and 17.3 per cent. on second mining; 13 per cent. of the breaker production was used for boiler fuel. The prepared yield for 28 producing years averaged 57.7 per cent.

Colliery B is in the main southern basin, and the coal lies in several local basins. No towns or streams requiring surface support lie over the coal. The measures are broken as a result of several reverse strike faults, and crop twice. The coal is more friable than that of the western middle field. It is very carefully mined to obtain a maximum yield from the seven seams, which are being mined. The general dip of the developed territory is 50°. The beds do not have big intervals between them; the thickest bed averages 10 ft., the smallest 3 ft. The boiler fuel used was 13.6 per cent. of the coal produced by the breaker. In 1923, the yield of prepared sizes was 46.6 per cent. The colliery is not exhausted, the figures given being from certain areas studied.

Colliery C is near Hazleton, and five seams, including the Mammoth, were mined; the others were much thinner. A small stream, a road, and a railroad cross the property, but they did not interfere seriously with the mining, as the stream was flumed and the railroad brought back to grade after subsidence took place. The coal, except where faulted or badly folded, dipped 25° to 45° and laid in two basins, one being small and shallow. The main basin had a maximum depth of 500 ft. below the surface. Generally speaking, the mining conditions were very good, the coal was hard and the yield, in prepared sizes, was 64 per cent. Several strippings helped to increase the recovery from the property. The management for the last 30 years had largely a robbing proposition and, by careful work, succeeded in removing the pillars without bringing on a squeeze. The average boiler fuel consumption for over 20 years was 17.3 per cent. of the breaker product. The colliery is considered exhausted and has been abandoned.

Collieries D and E are in built-up sections of the Wyoming territory and under the Susquehanna River wash. Colliery D was abandoned when the Susquehanna River broke into the mine following a squeeze that damaged surface property. The seams are quite regular, with an occasional thinning, which had no serious effect on mining. The section through the colliery shows the limbs of an anticlinal cropping in the river wash, which reduced the workable area of the three upper seams. Colliery E was abandoned because of damage to surface property and high pumping costs. The property was well silted in certain areas, nevertheless subsidence occurred, which eventually forced the abandonment of the colliery. Eight beds were mined; the smallest was 4.5 ft. thick and the largest 20 ft. The maximum pitch was about 12°. Accurate records of boiler fuel used were not kept, but it was probably 8 per cent. of the breaker production. Neither of these collieries can be considered as exhausted as it is possible that additional coal may be recovered from both, but the risk is great and the cost will be very high. What has happened to these collieries may happen to others in the lowlands of the Wyoming valley.

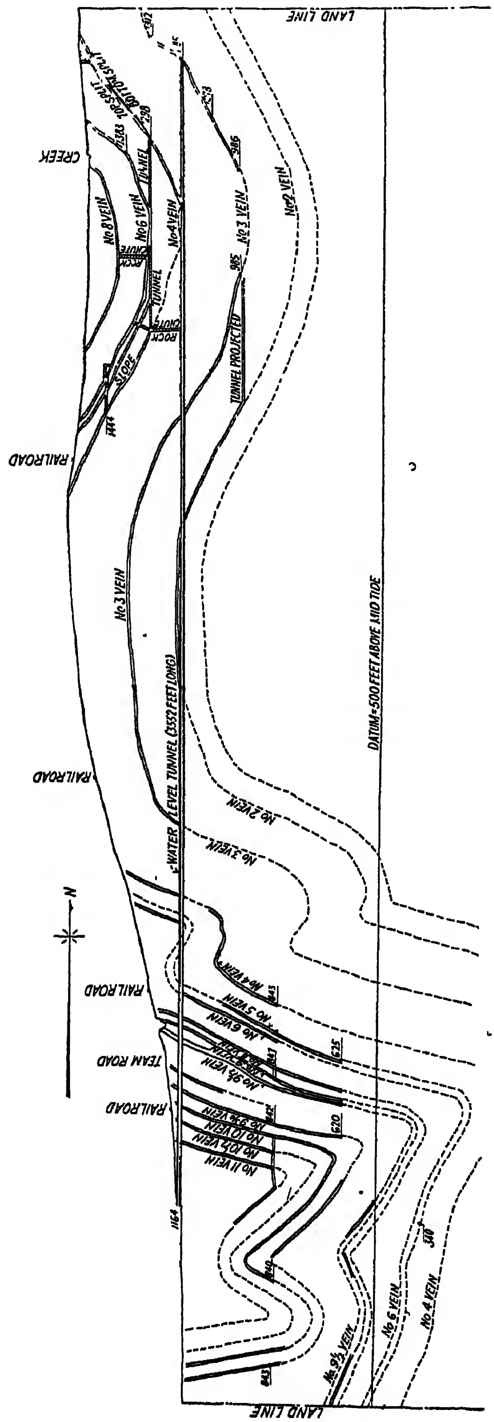

FIG. 3.—TYPICAL CROSS-SECTION THROUGH COLLIERY A.

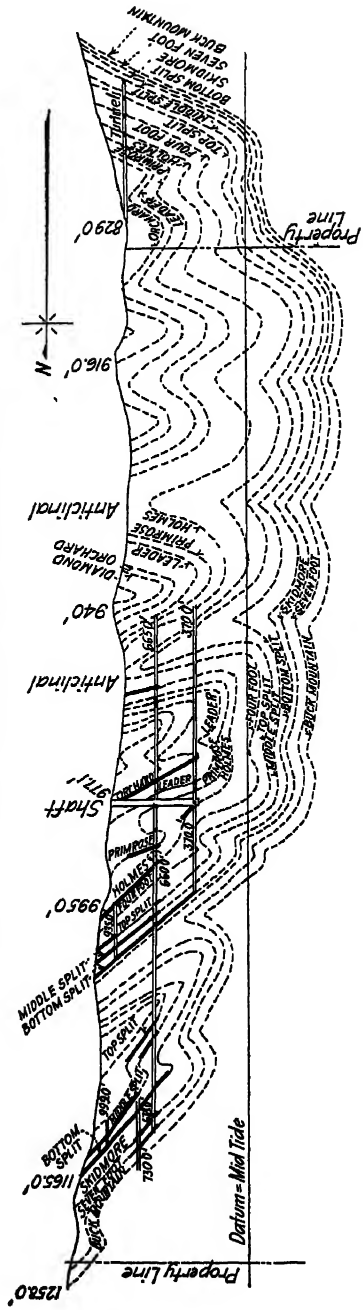

FIG. 4.—TYPICAL CROSS-SECTION THROUGH COLLIERY B.

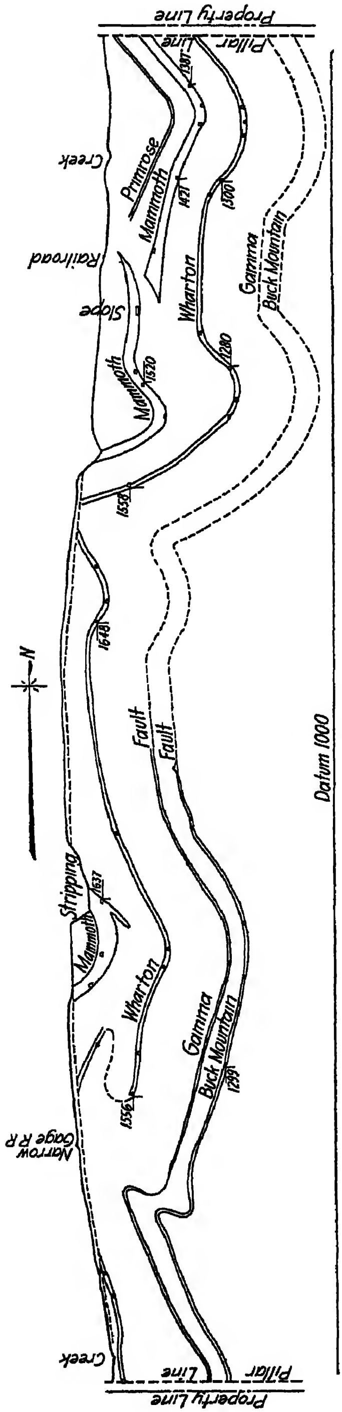

Fig. 5.—Typical cross-section through Colliery C.

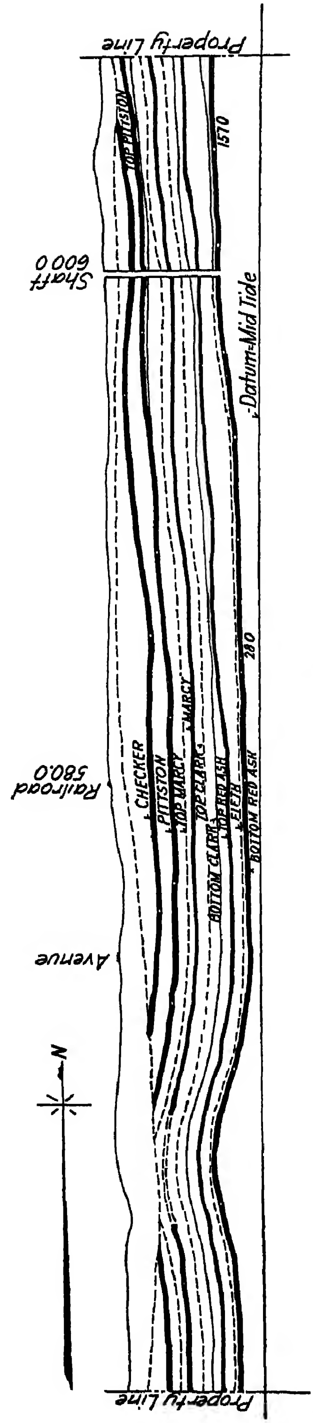

Fig. 6.—Typical cross-section through Colliery D.

Probably we all have seen areas of beds that were being robbed practically clean. Three of the collieries have been mentioned to emphasize the fact that, as an overall proposition, the percentage of recovery for the average colliery is not as high as is generally supposed.

Conclusions

We were unable to derive empirical formulas from a large volume of information, because records have not been kept in a manner that would permit them to be combined with records from other collieries with any assurance of accuracy.

The recovery percentages from limited areas give a distorted conception of results to be obtained from an entire colliery, as is shown from the results obtained from the five collieries described.

To determine the recovery from individual seams of coal, accurate records of coal produced from each breast and line of pillars must be kept; also accurate surveys must be made and bed sections must be taken at frequent intervals. If it is not advisable to keep records of output by breasts, the record of output from sections of gangways should be kept.

This paper is presented with the hope that it will stimulate a greater effort to conserve our anthracite, the making of good surveys, and the keeping of good records so that the life of the individual mine will not be unnecessarily shortened.

The author wishes to acknowledge the assistance and information received from members and friends in the various fields, and also to extend his thanks to Mr. R. G. Pfahler for the preparation of the curves and general compilation of the data.

DISCUSSION

R. V. Norris, Wilkes-Barre, Pa.—I would criticize the wording in one particular: the author does not make clear his use of recovery. It is, I think, the general practice of the region to distinguish between the words "removal" and "recovery," removal being the coal taken from the mine, and recovery the coal shipped to market.

The five collieries selected, with the possible exception of C, are not exhausted, and it may interest the author to learn that within the last few days a lease has been made for the coal remaining in colliery C, which it was thought had been exhausted. The figures given are, of course, on old collieries; they do not reflect the present recoveries. A small basin in the Cooper seam, mined by the Alden Coal Co., under about 125-ft. cover within the past few years actually yielded a recovery to market of over 75 per cent. of the original coal in the ground; that is under the most favorable conditions. It was a very pure seam under light cover, with no geological difficulties, no faults, a very regular condi-

tion all the way through, but that is possibly very close to the maximum that can be hoped for.

I believe, however, that at the present time under reasonable conditions a removal from new work of from 75 to 80 per cent. can be expected and recovery of from 60 to 65 per cent., depending on conditions. The recovery is lower in the softer and more crushed measures of the southern field than in the harder coals of the northern field.

S. A. Taylor, Pittsburgh, Pa.—Have you any figures that will show the relative recovery of the different pitches of the seams? One of the slides showed a seam that was badly faulted; I can easily understand that the recovery there would be very small on account of the low pressure.

Henry H. Otto.—The recovery in those conditions in flat seams is somewhere between 70 and 85 per cent. for the smaller seams; in the heavy pitching seams, where the coal is much softer, a recovery of about 60 to 65 per cent. is almost as good as one can expect. In the northern field about the same percentages hold true as in the southern field under the same conditions, except for the change of pitch. We think that with a recovery of 60 to 65 per cent. we are doing exceedingly well.

Douglas Bunting, Wilkes-Barre, Pa. (written discussion).—The statement that mining, in the early days, was most skillfully done for the reason that later attempts to rob these areas revealed that the pillars were reduced to a minimum for required support is a misconception. The mining of early days was at comparatively small depths below the surface; as a result the problem of adequate pillar support was of little moment or concern. For illustration, veins varying from 6 to 16 ft. in thickness can safely be mined to a depth of 150 ft. below the surface with chambers 24 ft. wide on 30-ft. centers, or these same veins could be mined to a depth of 300 ft. with the same width of chambers on 35-ft. centers.

The statement that many squeezes were caused, not by careless engineering and mining, but by unknown geological conditions that became known after mining had progressed, is apparently placing the responsibility on the laws of gravitation; whereas many of these failures were due to following the practices and methods of mining in vogue at shallower depths.

The marketable recovery from the five collieries cited is interesting information and proves, in a measure, that estimates of ultimate recovery are frequently excessive. However, the collieries cited may not be absolutely exhausted and the methods and practices pursued in the mining may have been erroneous, whereas under present day practices greater ultimate recoveries would probably have resulted. It is unfortunate that records of mining in the past have not been kept in a manner that will permit determining the ultimate recovery from individual seams, but such records are being kept now by some companies and much can be accom-

plished in the determination of ultimate recovery by tabulating the results of mining in individual seams under various conditions.

GRAHAM BRIGHT, Pittsburgh, Pa. (written discussion).—The statement that boiler fuel is roughly 10 per cent. of all the coal mined in the anthracite region, and that it is generally regarded as a loss, would seem hardly fair to the mine producing its own power. If power is purchased, all the coal would be shipped and there would be no loss, notwithstanding the fact that the power cost may be the same in each case.

The average production of anthracite during the last four or five years is about 90,000,000 tons per year. With 10 per cent. used for boiler fuel, this would represent 9,000,000 tons of coal, most of which is merchantable.

If we assume that the average power consumption in the anthracite field is 15 kw.-hr. per ton, the total amount of power required per year would be 1,350,000,000 kw.-hr. If all of the power required was produced in two or three large central stations, a large saving in fuel could be effected. In a station of this type a kilowatt-hour should be produced with about 2½ lb. of coal. On this basis, the total amount of coal required for power for one year would be about 1,700,000 tons. As at present about 9,000,000 tons are used for boiler fuel, the saving effected by electrifying all mines and purchasing power from large central stations would be about 7,300,000 tons. The cost of electrification would be high, but the saving would be such that the electrification would soon pay for itself.

HOWARD N. EAVENSON, Pittsburgh, Pa. (written discussion).—The curves showing the trend of variation in recovery confirm the fact stated by Clagett, in "Systems of Mining in Pocahontas Coal Field and Recoveries Obtained," that the recovery from thick seams is not as great as from thinner ones, although the problem is a much simpler one in the latter case than in the one under discussion, because of the lack of complicating features and the uniform character of the seam.

It is questionable, to my mind, whether the amount of coal used at the plant as fuel should be classed as a loss in mining. It would seem better to include all the coal produced, and not only that shipped, in calculating the percentage of coal recovered. Whether power is purchased or not, the efficiency of boiler plant and machinery, the amount of water encountered, etc., all enter into the amount of fuel coal used, while only the latter has any appreciable effect on the recovery in mining. It is to be hoped that more data of this kind from the anthracite region will be available later.

J. B. WARRINER, Lansford, Pa. (written discussion).—The data obtained, while by no means complete, is conclusive evidence of the fact that the ultimate recovery in anthracite mining is less than engineers

generally have been willing to admit. It must be remembered that the properties on which figures are obtainable have been mined over a long term of years, during which time mining methods were admittedly more primitive and less efficient than they are at the present time. In other words, an operation starting in a virgin territory at this date and worked to exhaustion under present-day methods might show greatly better results in total recovery. Nevertheless, the figures given should be a desirable shock to all mining men in the anthracite region, and should lead to a careful study of possibilities of improving the record as shown.

Mining a Steeply Pitching Anthracite Vein by Successive Skips

BY J. S. MILLER, LANSFORD, PA.

(New York Meeting, February, 1925)

THIS paper describes the method of mining a steeply pitching anthracite seam on a heavy pitch in the Orchard vein in No. 1 Tunnel of the Lehigh Coal and Navigation Co.

The Orchard vein in the Nesquehoning district has a thickness of about 15 ft. on the level of the gangway, which is sufficient thickness for the gangway to be driven full width in coal. Above the gangway, the vein becomes thinner and near the surface is about 8 ft. thick. The coal is of good hard quality and carries a fairly good roof. The pitch of the vein is between 65° and 70°.

Usually the breast-and-pillar method has been employed in the mining of the steeply pitching veins in this section. This method produced good results as far as the mining of the breast was concerned but the recovery of the pillar was difficult as to full extraction and expensive as regards costs. To recover these pillars after the breasts were driven, three methods were commonly employed; they were known as the pillar-breast method, pillar-skipping method, and pillar-chute method.

The pillar-breast method consisted in putting a battery in one corner of the pillar (Fig. 1), coupling the manways *A* and *B* of the two adjacent breasts for two or three lengths ahead of the working face of the pillar being cut, and working the pillar as a breast. This method was used by the more skilled miners of the earlier days but has practically been abandoned. A slight variation of this method is used to some extent. In this case, the pillars are left wider than ordinarily and of sufficient width for the full-width breast to be driven in and a small pillar left on each side of the pillar breast between it and the regular breasts. With the completion of the "pillar breast," the ribs are drilled full of holes, loaded, and fired.

In the pillar-skipping method (Fig. 2), the middle breast of three is not driven through and is drawn empty while the outside and inside breasts are kept full. At the corners of the two pillars standing, batteries are put in and a skip of the two pillars started; the coal in the two pillars is blasted over into the empty breast. A small pillar is kept along the

manways of the two skips between the skip manways and the manways of the two full breasts. When the skips are completed, these two small pillars are drilled full of holes, loaded, and fired.

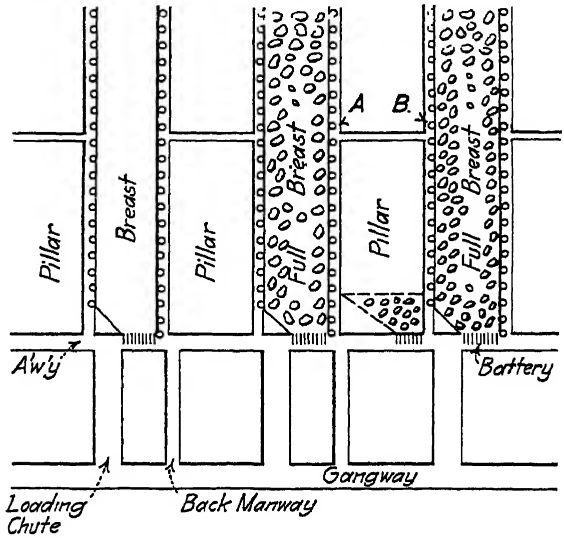

FIG. 1.—ADJOINING BREASTS LEFT FULL; BATTERY PUT IN AT CORNER OF PILLAR; PILLAR CUT ACROSS OLD MANWAYS *A* AND *B*; KEPT COUPLED THREE LENGTHS AHEAD OF WORKING FACE IN PILLAR BREAST.

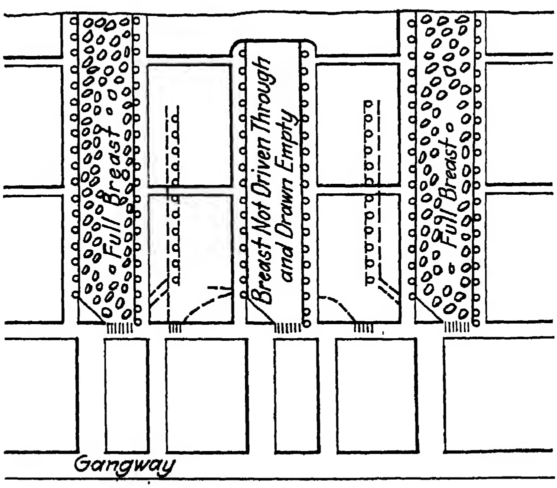

FIG. 2.—MIDDLE BREAST NOT DRIVEN THROUGH AND DRAWN EMPTY; TWO BATTERIES PUT IN AT LOWER CORNERS OF PILLARS AND SKIP CARRIED UP; TWO PILLAR SKIPS DUMPED INTO EMPTY BREASTS.

The pillar-chute-mining method (Fig. 3) is the method most commonly employed. A chute is driven up the center of the pillar for the entire length of the pillar, which is then robbed down by means of this chute.

Two miners cut this pillar, starting at its top and retreating as the pillar is cut. The chute is used for traveling by the men as well as for running down the coal. This method is very expensive, as the labor cost of

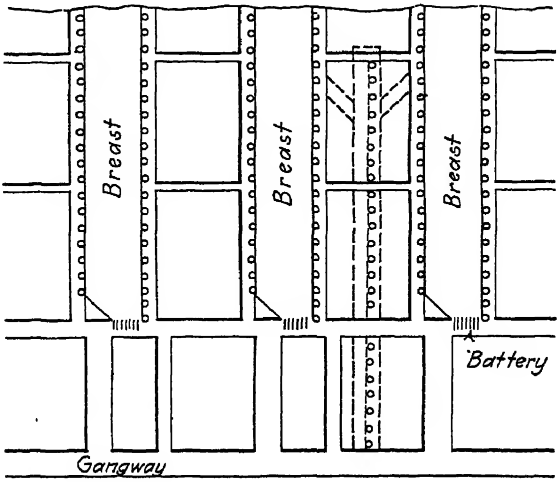

FIG. 3.—CHUTE DRIVEN UP CENTER OF PILLAR; SIDE CHUTES DRIVEN OFF EVERY 20 TO 30 FT.; PILLAR CUT AND COAL RUN DOWN PILLAR CHUTE.

driving the chute plus the price paid for the cutting of the pillar is approximately double that paid for the driving of a breast of the same width as the pillar. In addition, the expense of maintaining this chute is great,

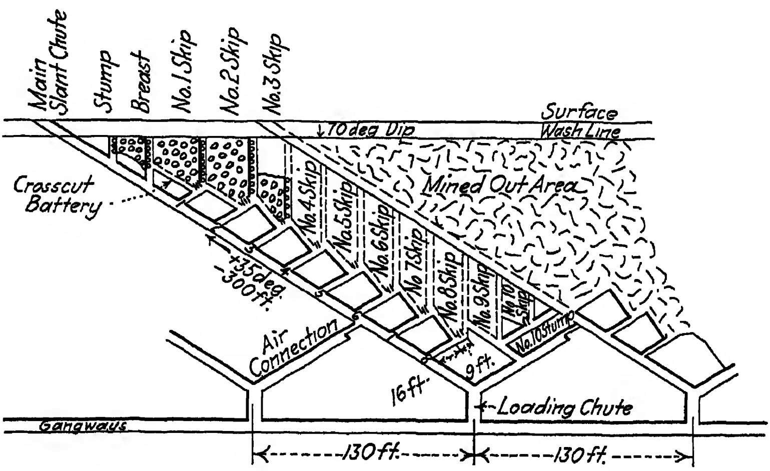

FIG. 4.—PRESENT METHOD OF MINING THESE VEINS.

because on this heavy pitch the chute is broken considerably by the coal running down it. Another objection to this method, and to an anthracite operator it is an important one for it is often the deciding

factor between profit and loss, is the very great breakage incurred in running the coal down a chute on this heavy pitch.

These three methods have been briefly described principally to give the reasons for adopting the method to be described. In this method (Fig. 4), chutes are opened on the gangway on about 130-ft. centers. Each chute, when opened off the gangway, is connected first with the preceding one by a slant chute, which serves to establish ventilation. When this is accomplished, the main chute is driven across the pitch on 35° to 45° and, while being driven, is ventilated by means of a small electric fan and air pipe through which air is conducted to the working face of the chute. This chute is driven through to the surface by two gangs of miners working double shift. By driving this chute across the pitch, the vertical lift is cut about in half, which aids materially in working the vein. These slant chutes will run about 300 ft. in length, and on account of the flatter pitch on which they are driven, than one driven directly up the pitch as in the pillar-chute method, they require comparatively little upkeep.

The first mining done off this slant chute is to open a breast. The upper manway of the breast will be about 25 to 30 ft. from the top of the hole to the surface. The manway of the chute under the breast is coupled and kept open to continue as an outlet for ventilation and to serve later as a means of running the coal in the stump between the upper breast manway and the surface when this stump is cut in the robbing of the chute. The length of this breast will be about 50 ft. While the breast is being driven, the first auxiliary chute is driven about 20 ft. below the lower breast manway and in the opposite direction to the main slant chute for about 25 ft. so that a pillar of from 25 to 30 ft. is obtained between the breast and the top of this auxiliary chute. In this auxiliary chute, a check battery is put in and a crosscut driven from it to the lower breast manway, which establishes ventilation for the driving of No. 1 skip. When the breast is completed, skip No. 1 is started from the top of No. 1 auxiliary chute and cut over to the breast. This manway and skip are continued through to the surface.

While No. 1 skip is being driven, No. 2 auxiliary chute is started and driven and a battery put in this chute. A crosscut is driven from the battery to the manway of No. 1 skip. If No. 1 skip is not completed when No. 2 auxiliary chute is ready for No. 2 skip to be started from its top, No. 3 auxiliary chute is started below No. 2 auxiliary chute and made ready in a manner similar to Nos. 1 and 2. With the completion of No. 1 skip, No. 2 skip is started and carried through to the surface. In this manner, opening work for successive skips is always prepared in advance. Each skip increases in length and will reach a maximum at about the fourth skip, which runs into the main chute previously driven.

After several skips have been driven, the next main chute inside is probably driven and an air chute driven from it to the main slant chute

being robbed. When this connection is made and the skips have passed the point where this connection has been made, one of the two gangs of miners engaged in the cutting back of this chute cuts the stump at the top of the chute between the breast and the tip of the hole to the surface. The coal from the stump is run down the manway of the main chute, which was coupled together under the breast. After this stump is cut and the coal run out of it, the stumps remaining between the main slant chute and the work done above it are in turn cut and loaded. The stumping is continued down to the connecting air chute driven from the next inside main chute. It permits the mining from the next main slant chute inside. From this connecting air chute to the loading chute, a distance of about 80 ft., the stumps are left standing on account of the ventilation of the inside chute and as a protection to the gangway.

From seven to eight skips are taken on each main chute and usually two skips and a stump are taken on the air connecting chute between the main slants. Four or five skips can be left full of coal to serve as a reserve from which to draw while the men are driving another chute inside and getting ready to cut in; the driving and robbing of a chute usually requires from 11 to 12 months.

This method allows a complete extraction of the vertical pillar between these main slant chutes without any of the objectionable features of the pillar-breast method, the pillar-skip method, or the more expensive pillar-chute method. Four or five of these chutes are kept in continual operation, which furnishes work for four times this number of miners and permits concentration of working places.

Since the adoption of this method, the output has been increased from 40 to 90 cars per day and the cost per car has decreased approximately half. A test of cars loaded from this method of mining showed a percentage of prepared coal of 75.6 per cent., while a test of cars loaded from the pillar-chute method showed only 51.3 per cent. of prepared coal.

The extraction by this method is 68 per cent. with 32 per cent. remaining in the pillars and stumps.

The ventilation of this section is entirely by natural ventilation and has been a very simple problem, as the vein is practically free from gas and as each chute is being driven through to the surface, the air either goes up or down the chute depending on the outside temperature.

To ventilate a similar method of working when the lift is not being worked through to the surface or to an open gangway above and where the ventilation is furnished by a main ventilating air fan, it would probably be necessary to change this method somewhat. Under these conditions, more connecting air chutes would probably have to be driven between the main slant chutes. These could be driven, however, in such a manner as to serve as auxiliary chutes for use in driving the skips.

Simultaneous First and Second Mining on Steep Pitches*

By Dever C. Ashmead, Wilkes-Barre, Pa.

(New York Meeting, February, 1925)

Coal companies in the anthracite region are studying various methods of mining, seeking one that will shorten considerably the life of the gangway and thus decrease the maintenance charges. In steep pitching measures, the maintenance charges of the gangways are high because of the cost of replacing timbers and cleaning up falls. Therefore, if the length of the gangways can be decreased and the mining made more concentrated, the life of the coal gangways will be decreased and the maintenance charges per ton of coal produced will be smaller.

The practice in the anthracite region is to drive long coal gangways and to work the breasts as the gangway from which they were driven

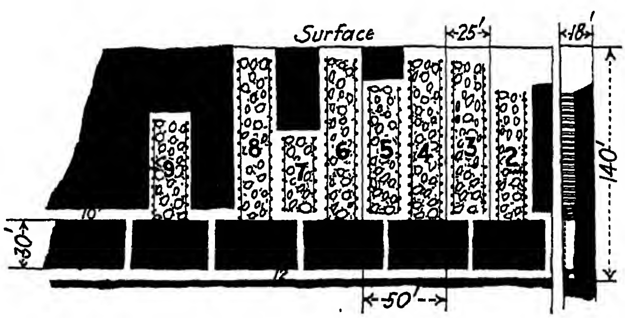

Fig. 1.—Cross-section of workings in Baltimore Bed, Wanamie Colliery.

is advanced. In many cases these gangways are from 4000 to 6000 ft. in length; and some are even longer. As these must be maintained not only until they have reached their limit but until second mining begins (which may be some years later), the maintenance charges are high. If the gangway is allowed to close, when the time comes to remove the pillars the cost of re-opening it may be as expensive as the original driving cost.

The Lehigh and Wilkes-Barre Coal Co., at its Wanamie Colliery at Wanamie, Pa., has been using a system of mining that, as far as the writer knows, is not used in any other place in the anthracite fields. The

* Published with approval of the Director of the Bureau of Mines.

work is being conducted in that section of the No. 28 Tunnel District that lies in the Baltimore bed.

Here, the Baltimore bed is 18 ft. thick and lies on a pitch from 60° to 80°; at one point at least, the pitch is 90°. Lifts are planned to be about 140 ft. apart vertically. The coal bed is to be divided into a number of sections along the strike, depending on the length of life that is found desirable for the gangways and also on the tonnage that is to be taken from this bed. The gangway is to be divided into sections 800 ft. long.

The main gangway is driven either in an underlying bed or in the rock. This is used as the main haulage road and at this colliery is in the Ross bed. Tunnels are turned from this gangway to the Baltimore bed; and the main gangway in this bed (12, Fig. 1), is driven about 12 ft. wide and 6 ft. clear of the rails. It must be well timbered; the sets are placed from 4 to 6 ft. apart and are fully lagged, particularly on the top and along the foot wall to prevent small pieces of coal and rock falling on the men. The Monkey Heading (10, Fig. 1), is driven 30 ft. above the main gangway; this provides for the air returns. Chutes are driven between the monkey heading and the main gangway at 50-ft. intervals. The monkey heading is about 5 ft. wide and 5½ ft. in height and is timbered in the same manner as the main gangway, but lighter timber is used.

Ventilation in the top lift of the colliery is natural; holes are put through to the surface as may be needed. The lower lifts will have their returns provided by driving a break through to the lift above.

Method of Mining

A breast is driven up 25 ft. wide at the extreme limit of the panel of the property. This breast goes up until it reaches either the surface or the gangway immediately above. Spaces are allowed for a manway on each side of the box 2, Fig. 1, which is about 16 ft. wide in the clear. Wherever possible the end of the box rests directly on the pillar between the monkey heading and the main gangway; this does away with the necessity of building batteries at the ends of the breasts.

Coal is slopped over, that is, the surplus coal which cannot be held in the box when mining, goes down the manway and is loaded through the adjoining chutes into the mine cars below and taken away; the rest of the coal is left standing in the box. While this breast is being driven, breasts 4 and 6 are being driven. In this work, there are usually employed three miners and three laborers, therefore it is desirable to have sufficient places for them all to work.

A pillar of coal is left between each of the breasts 25 ft. wide, thus making the breasts on 50-ft. centers. When breasts 2, 4 and 6 have been driven to the limit, breast 3 is driven in the pillar between breasts

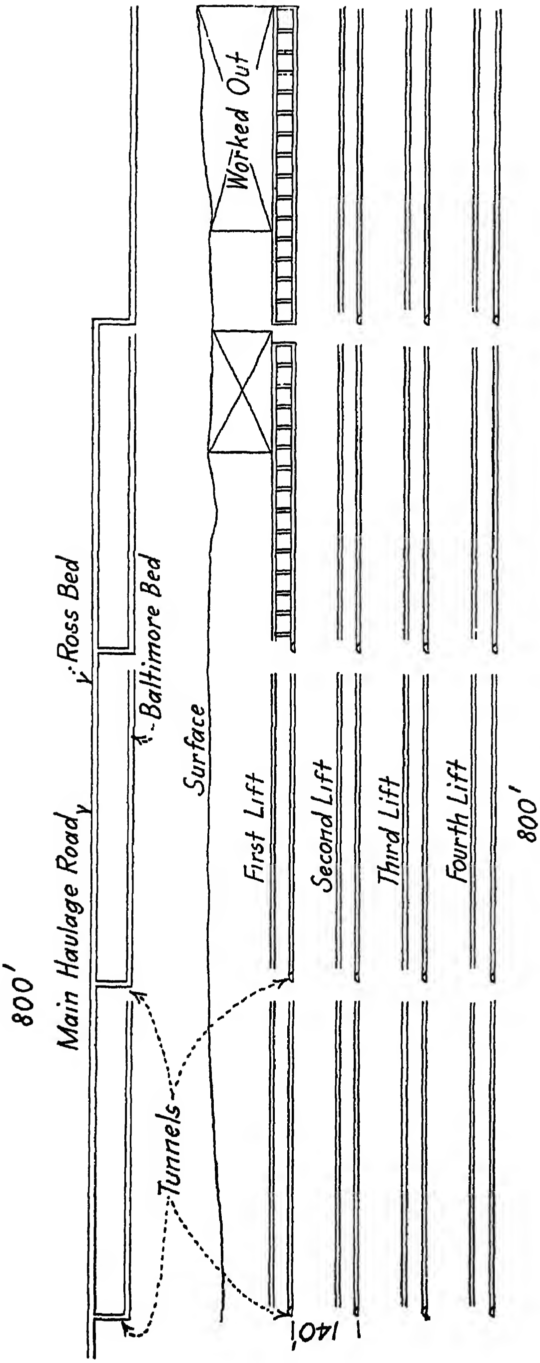

FIG. 2.—GENERAL PLAN OF LAYOUT OF WORKINGS OF WANAMIE COLLIERY.

2 and 4, and breast 8 is started. When breast 3 reaches its limit the coal is left standing in it, as in 2, 4, and 6, and another breast 5 is started in the next pillar; also breast 9 may be started.

When a breast is started in a pillar it is necessary to build a battery at the lower end of the box as the pillar is directly over the chute. This battery is braced by timbers that extend across the monkey heading. The battery extends the full width of the box, *i. e.*, 16 ft., as all of the pillar is taken out advancing. As breast 5 is being driven, coal is removed from the box in breast 2, and when breast 5 is driven to its limit, coal will be drawn from the box at breast 3, and breast 7 started when breasts 8 and 9 have been completed.

Before drawing coal from the box it is intended that there will be two full boxes of coal standing between the box that is being drawn and the place where the men are mining, thus giving them sufficient roof protection.

The three miners and the laborers work as a unit. That is, they plan the work so the coal can be drawn from a box as has been described, but they also have other places to work, in case there should be a car shortage or other troubles that might retard the drawing of coal from the boxes that have been completed.

The bed of coal is 18 ft. thick, and the breasts are driven in the lowest bench and are about 7 ft. high, therefore 11 ft. of coal is left standing in the bed.

When coal is finally drawn from the box, it is pulled back about 10 or 12 ft. at a time, then the miner drills holes in the roof or hanging wall and shoots the coal that has been left standing. This falls and is drawn down through the box. In this way all of the coal is recovered—about 40 per cent. when the breasts are driven and the balance when the boxes are drawn.

When box 2 is drawn, a slice is taken back with the box along the edge of the barrier pillar, because there is insufficient room to drive a breast between No. 2 and the barrier.

When the boxes are drawn all support to the roof will, of course, be removed. At present, one of these panels has been drawn back a distance of 800 ft., and the indications are that there has been no roof break, the top being a hard sandstone standing almost vertical. It is expected that when the roof breaks it will do so in large slabs, that will lean over and stay in position, therefore clogging the opening and tending to prevent loose rock dropping and mixing with the coal.

By this method very few breasts are left standing at one time, not over three, and, as stated before, the work is so planned that coal is left standing in only two boxes between the box being drawn and the place where the men are working.

As fast as the coal is drawn from the boxes the pillars between the monkey heading, No. 10, Fig. 1, and the main gangway, No. 12, are drawn back, thus rendering it unnecessary to maintain this part of the gangway. All the coal is continuously pulled back, leaving no old breasts and no old gangways. The panels are small, therefore the gangway is short and its life is short; consequently the maintenance charges are considerably reduced and the equipment is available for use elsewhere.

Ordinarily the posts for the boxes are placed on 6-ft. centers and heavily lagged; at times they must be placed closer. The posts must sustain considerable weight, particularly when the roof coal is being taken.

It is possible to secure about 45 tons a day from a set of breasts, therefore the number of panels required to produce a given output can be readily calculated.

The advantages accruing from this method of mining are many and important:

1. No coal is drawn from the boxes until work in the place is completed, consequently the coal packs tight in the box and takes the weight of the roof.

2. The coal is not left standing in the old chambers, for there are no old chambers, and roof falls are not apt to take place.

3. As the weight of the roof is taken by the box, it reduces the weight thrown on the coal at the face and reduces the tendency of the coal to run.

4. By not drawing coal from the boxes while driving the breast, roof falls in the breast are not liable to occur as continuous roof support is given; therefore roof rock is not mixed with the coal and the coal can be loaded into the mine cars clean when the box is drawn.

5. The life of a panel is short and therefore the maintenance charges are low.

There are probably some disadvantages, one of which is the degradation of the surplus coal the boxes cannot hold and which is sent down the manway. The indications from the car yield, however, are that the degradation is negligible.

Alabama Coal-mining Practice

By Milton H. Fies,* E. M., Birmingham, Ala.

(Birmingham Meeting, October, 1924)

Although pig iron from iron ore and red cedar charcoal preceded the mining of coal by many years, for tradition says that Alabama iron was used to shoe the horses of Andrew Jackson's soldiers, coal was not produced until 1830. In that year, coal was mined in Tuscaloosa County, in the vicinity of what is now the University of Alabama, and shipped on flat boats (obviously during the wet season, for there were many shoals) down the Warrior River to Mobile.

About that time, the first railroad, the Decatur & Tecumseh, was being constructed in northern Alabama to overcome the obstructions to freight traffic caused by the now well-known Muscle Shoals in the Tennessee River. This railroad was patterned after a railroad at Mauch Chunk, Pa., which was used for hauling anthracite from the Panther Creek valley. The Alabama railroad, which required about four years to complete, was 46 miles long and was used solely for the transportation of cotton. Although it was a failure, it demonstrated the practicability of the undertaking, and the succeeding railroads, which were begun in 1834, 1848, 1852, and 1853, added immeasurably to the development of the coal resources of the state. Between the years 1830 and 1860, practically all the coal mined in the state was dug from the outcrop near the river or from the river beds, and shipped by boat to Mobile. Transporting coal down the Warrior River was an undertaking of great risk, required much skill, and frequently had heavy losses. It is recorded that this coal was sold to the gas company in Mobile in 1844; it was most likely mined from the Black Creek Seam.

In 1914, on the completion of Lock 17, the Warrior River became a navigable stream from 9 miles above Cordova, Ala., to Mobile and offers a great opportunity for the future development of the coal resources of the state.

Coke for foundry use was first made in Alabama in 1855. It was 21 years later, however, in 1876, that pig iron was made from Alabama ore with Alabama coal.

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* Vice-President in Charge of Operations, DeBardeleben Coal Corpn.

Coal and ore mines in Alabama during the "War between the States" aided the South materially and caused great concern to the Federal Government until Federal troops, in 1865, captured the iron works and coal mines. This war awakened the nation, and the South especially, to the possibilities of Alabama's resources. Since that time, notwithstanding difficulties and obstacles, the growth has been most marked.

The perfection of the process of the manufacture of open-hearth steel from Alabama pig iron, on Thanksgiving Day, Nov. 30, 1899, marked the beginning of substantial development in Alabama. The production of coal for the 24 years, 1900–1923, inclusive, was 355,583,769 tons; the production for the preceding 24 years was 74,329,369 tons. To the perfection of the open-hearth process, no small amount of this increase is due. After the consummation of this process, large producing companies that were making pig iron began comparing the coal and ore resources and, in the main, adopted the policy of withdrawing coal from the commercial markets.

Ownership

The ownership of coal seams in Alabama is mainly held through fee or mineral rights, although a number of operations are on royalty bases. The royalty paid varies from 5 to 7 cents per ton on the low-grade, high-ash coals, to 15 to 20 cents per ton for the best grade coals. The nature of the ownership of coal lands determines, to a considerable extent, the percentage of recovery from a given area. As it has been found costly to mine pillars under land where the surface is owned by individuals, the size of the pillars left to protect surface is of prime consideration. Some coal companies in this state leave large pillars between rooms, which can be most easily recovered by robbing, when the ownership is in fee; but when such companies are in possession only of the mineral rights, pillars only of sufficient size to support the surface are left, and consequently are lost. The practice of longwall or semi-longwall mining is prohibitive where the surface is not owned by the coal company.

Prospecting

Prospecting in Alabama, where the coal lies close to the surface, is performed mainly by means of test pits on the outcrop, at intervals of about ¼ mile, and then tieing up these prospect pits with levels. Where the acreage is large, it is not uncommon to put down diamond-

drill holes into the body of the property, even though coal along the outcrop persists in thickness, quality, and regularity as to elevation. When it is planned to develop a coal property, with only the coal showing on the outcrop and then pitching into the body of the property, prospecting is done along the outcrop and, where the pitch is not too great or the basin shown by geological study not too deep, drill holes are put into the body of the property. Where the depth of the coal prohibits diamond drilling, geological study of the strata above the coal is made to determine the probable persistency of the seam.

In a region where the coal seam changes, both as to direction and character, without warning, and in comparatively short distances, this latter method is not satisfactory and development following such prospecting involves great risk. In one case where seven drill holes were put into a body of coal, consisting of 750 acres, and from twelve to fifteen test pits were opened on the outcrop, two faults were encountered in the mining that the prospecting did not reveal.

In the Cahaba field, where the coal lies pitching and in basins, geological study and test pits on the outcrops have been the only means of prospecting. When this field was first developed, many operators came to grief; it is only within recent years that a knowledge gained from the actual mining of coal in these basins has furnished tangible information and the risk has been eliminated.

Statistics

Table 1 shows production of coal mined in Alabama from 1870 to 1923, the number of men killed, the number of tons produced per life lost, the number of employees per life lost during the same period, and the tons of coke produced annually from 1880 to 1923, inclusive. Table 2 shows the tons of coal produced from the various seams, according to counties, for the Warrior, Plateau, and Coosa fields for 1923. Table 3 shows the production from the Cahaba field, according to counties and seams, for 1923. Table 4 shows the average number of days coal mines worked in the ten most important coal-producing states, according to compilation of the United States Geological Survey, from 1917 through 1921. Table 5 shows average analyses, as furnished by the Bureau of Mines, covering all seams mined in Alabama, together with the fusing temperatures of the ash.

Fig. 1 illustrates typical sections of thirty seams mined in the state. The production of the mines is used as follows: Railroads, 30 per cent., other steam users, 13 per cent., furnace companies, 43 per cent., domestic, 10 per cent., bunker, 4 per cent.

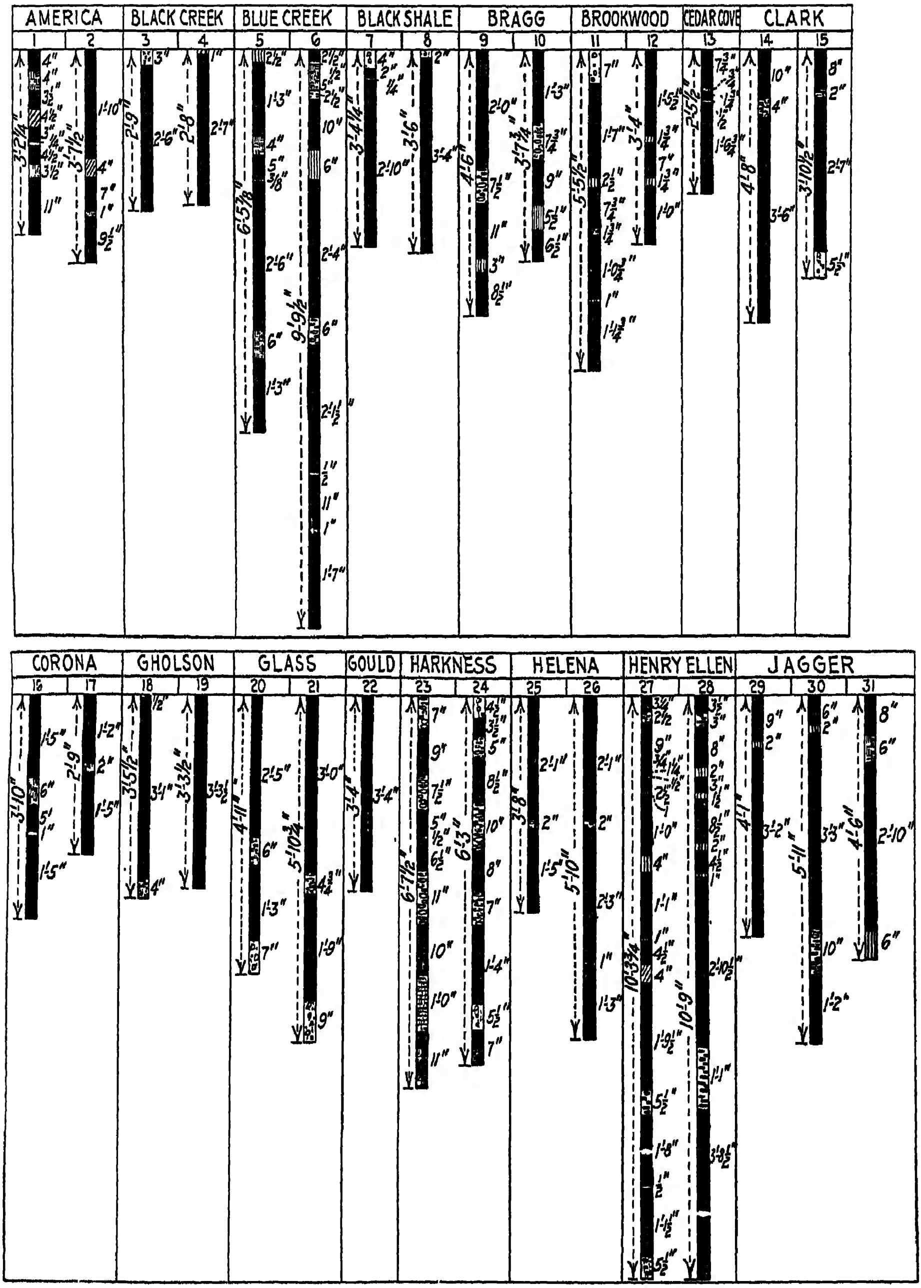


FIG. 1.—TYPICAL SECTIONS OF 30 SEAMS MINED IN ALABAMA.

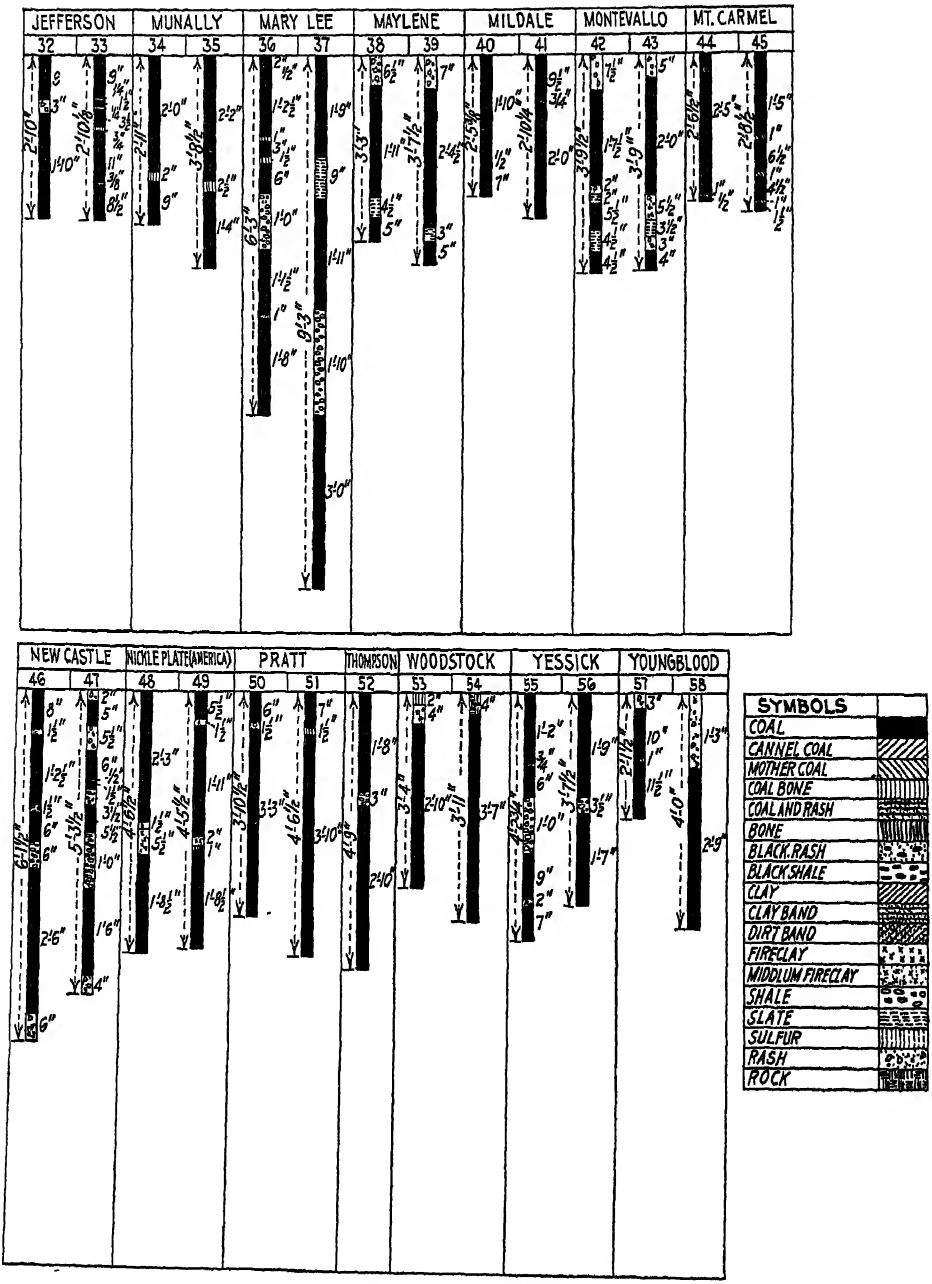


FIG. 1.—(*Continued*).

TABLE 1.—*Coal and Coke Production in Alabama*, 1870–1923

Year	Coal	Number of Men Killed	Tons Produced per Life Lost	Number Employees per Life Lost	Coke
1870	13,200				
1871	15,000				
1872	16,800				
1873	44,800				
1874	50,400				
1875	67,200				
1876	112,000				
1877	196,000				
1878	224,000				
1879	280,000				
1880	380,000				60,718
1881	420,000				109,033
1882	896,000				152,940
1883	1,568,000				217,531
1884	2,240,000				244,009
1885	2,492,000				301,180
1886	1,800,000				375,054
1887	1,950,000				325,020
1888	2,900,000				1,030,510
1889	3,572,893				508,511
1890	4,090,409				1,072,942
1891	4,750,781				1,282,496
1892	5,529,312				1,501,571
1893	5,270,042	17	310,000	529	1,168,085
1894	4,361,312	19	229,200	470	923,817
1895	5,705,713	38	150,150	217	1,444,359
1896	5,745,617	28	205,201	353	1,479,437
1897	5,893,771	38	151,122	326	1,443,017
1898	6,466,741	45	143,705	220	1,663,020
1899	7,484,778	40	187,119	323	1,787,809
1900	8,273,362	37	223,603	386	2,110,837
1901	8,970,617	41	218,186	345	2,148,911
1902	10,329,479	50	206,590	358	2,552,246
1903	11,700,753	57	205,276	340	2,693,497
1904	11,273,151	84	135,204	210	2,340,219
1905	11,900,153	185	64,325	93	2,576,786
1906	12,851,775	96	133,873	101	3,217,068
1907	14,424,863	154	93,668	131	3,096,722
1908	11,523,299	108	106,697	174	2,335,602
1909	13,790,268	129	106,901	156	3,047,510
1910	16,139,228	238	67,812	92	3,231,399
1911	15,011,853	209	71,827	108	2,756,697
1912	16,513,040	121	136,471	193	2,881,861
1913	17,907,284	124	144,413	203	3,531,505
1914	15,525,903	126	123,222	190	3,092,771
1915	15,266,831	63	242,331	366	2,958,062
1916	18,234,625	118	154,531	214	4,385,493
1917	20,413,811	108	187,298	256	4,868,598
1918	19,521,840	110	177,473	239	4,344,726
1919	15,928,196	93	171,277	276	3,397,748
1920	17,391,437	78	222,967	352	3,982,472
1921	13,015,017	80	162,688	334	2,547,664
1922	18,757,681	183	102,501	160	3,760,004
1923	20,919,303	89	235,048	183	4,689,641

TABLE 2.—*Production, in 1923, from Different Seams of Warrior, Plateau, and Coosa Fields, According to Counties*

Name of Field and Seam	Blount County		Cullinan County		Jefferson County		Marion County		Tuscaloosa County		Walker County		Winston County		Etowah County		Total Number of Mines	Total Production, Tons
	Number of Mines	Production, Tons	Number of Mines	Production, Tons	Number of Mines	Production, Tons	Number of Mines	Production, Tons	Number of Mines	Production, Tons	Number of Mines	Production, Tons	Number of Mines	Production, Tons	Number of Mines	Production, Tons		
Warrior field																		
America........					12	373,362					7	385,716					19	759,078
Black Creek.....	1	221,207	1	16,526	9	496,021	4	184,332			12	736,224	3	9,400			30	1,663,710
Blue Creek......					3	59,438			2	227,132							5	286,570
Brookwood......									10	473,457							10	473,457
Carter..........									5	86,610							5	86,610
Corona.........											5	476,138					5	476,138
Jagger..........											14	1,007,118					14	1,007,118
Jefferson........					5	363,697					1	1,950					6	365,647
Mary Lee.......					19	3,705,812					24	2,399,598					43	6,105,410
Milldale........									6	414,619							6	414,619
Mt. Carmel....											21	830,017					21	830,017
Pratt...........					28	4,910,951					21	400,337					49	5,311,288
Total.........	1	221,207	1	16,526	76	9,909,281	4	184,332	23	1,201,818	105	6,237,098	3	9,400			213	17,779,662
Plateau field																		
Underwood.....		163,231													2	9,905	Several small mines	173,136
Coosa field																		
None active.....																		

Total production from all beds in active coal fields 20,919,303 tons.

Total number of mines, as reported, 272 (in a number of cases several openings were considered as one mine).

Total number of active seams, as reported, 29.

TABLE 3.—*Production in 1923, from Different Seams of Cahaba Field, According to Counties*

Name of Seam	Bibb County		Shelby County		St. Clair County		Tuscaloosa County		Jefferson County		Total Number of Mines	Total Production, Tons
	Number of Mines	Production, Tons	Number of Mines	Production, Tons	Number of Mines	Production, Tons	Number of Mines	Production, Tons	Number of Mines	Production, Tons		
Black Shale			1	21,210							1	21,210
Bragg									2	122,642	2	122,642
Clark	2	155,642	1	43,903							3	199,545
Gholson			2	79,368							2	79,368
Glass									1	200,461	1	200,461
Gould									1	800	1	800
Harkness					3	357,024					3	357,024
Helena			7	235,991	1	50,396					8	286,387
Henry Ellen					3	392,360			1	14,858	4	407,218
Montevallo			3	122,722							3	122,722
Thompson	9	365,784	2	88,075							11	453,859
Nunnally									4	35,182	4	35,182
Wadsworth			2	3,253							2	3,253
Woodstock	7	471,069					1	150			8	471,219
Yessick	1	700									1	700
Youngblood	2	204,915									2	204,915
Total	21	1,198,110	18	594,522	7	799,780	1	150	9	373,943	56	2,966,505

TABLE 4.—*Average Number of Days Coal Mines Worked per Year, from 1917 to 1921, in the Ten Most Important Coal-producing States, According to Compilation of the United States Geological Survey*

State	1917, Days Mine Worked	1918, Days Mine Worked	1919, Days Mine Worked	1920, Days Mine Worked	1921, Days Mine Worked
Alabama	273	278	239	247	166
Colorado	263	255	225	255	164
Illinois	243	238	160	213	152
Indiana	221	227	148	192	128
Kansas	216	234	182	204	137
Kentucky	214	230	189	182	152
Ohio	210	223	164	188	134
Pennsylvania (bituminous)	261	269	218	244	151
Tennessee	241	265	201	234	154
West Virginia	225	238	200	198	149

The average distribution by states, of Alabama's production of coal is:

	PER CENT.
Alabama	67.0
Arkansas	0.6
Florida	1.8
Georgia	10.9
Louisiana	8.0
Mississippi	9.5
South Carolina	0.1
Tennessee	1.5
Texas	0.6

About 60 per cent. of the coal produced is put through a cleaning process, for which washeries and mechanical cleaners are extensively used. The bulk of the coke output produced is used principally in the blast furnaces of the state.

Estimates of the ore and coking coal reserve must of necessity be an approximation. A fair estimate of the red-ore reserve has been placed at 1,500,000,000 tons; the brown-ore at 75,000,000 tons. If the high-silica red and brown ore can be rendered available for smelting through concentration, it is probable that the total reserve would be increased by fully 1,000,000,000 tons. However, a method of concentration has not yet been devised.

The estimated tonnage of coking coals in the Warrior field is given at 4,195,328,000.[1] While the coking-coal reserve, which does not include

[1] Ernest F. Burchard and Charles Butts: Iron Ores, Fuels, and Fluxes of the Birmingham District, Ala. U. S. Geol. Surv. *Bull.* 400 (1910).

coals in the Cahaba field, and which probably will be used for coking, is in excess of the iron-ore reserve, one must not lose sight of the fact that the ownership of the ore is limited principally to a few iron-producing companies, whereas the ownership of the coal is in many individuals. The total estimated coal reserve in Alabama, at the end of 1920, is placed at about 67,000,000,000 tons by Marius R. Campbell, of the U. S. Geological Survey.

About 77 per cent. of the coal of the state is produced with permissible explosives and 23 per cent. with black blasting powder; Alabama was one of the pioneers in the use of permissible explosives. About 50 per cent. of the coal mined is undercut with machines.

Investigations made for the U. S. Coal Commission showed that the average total recovery from all seams mined in Alabama has been only 68 per cent., after allowance was made for all losses. At some few mines, in the Cahaba field, under extremely bad top conditions, the recovery has been only 50 per cent. The percentage of recovery is steadily increasing in a number of the mines, especially in mines of the large producing companies, as the result of improved systems of mining.

The situation, with reference to the best grade coals used for domestic purposes, is not unlike the conditions in the anthracite fields of Pennsylvania. In Alabama, the highest grade coals sold for domestic purposes occur in seams 36 in. or under, lying flat, or in thicker medium or steeply pitching seams with bad top, gas and much water. The slack coal from these seams (which produce the lump and other domestic sizes), which varies from 40 to 80 per cent. of the total tonnage, must be sold in competition with the steam run-of-mine coals produced from the thicker low-grade coals. As a result, the slack obtained from the high-grade coals is sold at a loss and the lump and egg coals are sold at the higher prices. The producer of domestic coals, therefore, tries to mine as much of the larger sizes as is possible.

The Coal Fields

The coal fields of the state are the Warrior, Cahaba, Coosa, and Plateau; see Fig. 2. More than 99 per cent. of the coal is obtained from the Warrior and Cahaba fields, while the Warrior produces from six to seven times as much as the Cahaba.

The Warrior coal field is located north and west of Birmingham and comprises parts of Jefferson, Winston, Marion, Fayette, and Tuscaloosa Counties, and the entire Walker County. Its known area is about 3000 sq. mi. According to Henry McCalley, formerly assistant state geologist: "There is no way of telling how far its strata may extend to the southwest and west, under the overlying Cretaceous and Tertiary formations." There are six groups of seams, making in all twenty-three regular coal seams. By the term "regular" is meant regularity as to

TABLE 5.—*Average Analyses and Fusing Temperatures of Ash from Alabama Coal Seams**

Seams	Moisture, Per Cent.	Volatile Matter, Per Cent.	Fixed Carbon, Per Cent.	Ash, Per Cent.	Sulfur, Per Cent.	B.t.u.	Fusing Temperature of Ash, Degrees F.		
							Minimum	Maximum	Average
America	0.94[a]	30.06[a]	60.30[a]	8.69[a]	1.79[a]	13,872[a]	2,365	2,855	2,570
	2.61[b]	29.56[b]	59.27[b]	8.55[b]	1.76[b]	13,591[b]			
Black Creek	1.28	34.18	61.03	3.49	1.04	14,521	2,010	2,800	2,430
	3.41	33.39	59.71	3.41	1.02	14,237			
Black Shale	1.50	36.20	59.80	2.50	0.60	14,670	2,060	2,110	2,080
	4.60	35.10	57.90	2.40	0.60	14,210			
Blue Creek	0.77	24.40	65.04	9.78	0.72	13,912	2,620	2,910	2,830
	3.84	23.64	63.01	9.48	0.70	13,478			
Bragg	0.80	34.20	58.20	6.80	1.50	14,070	2,150	2,180	2,160
	2.30	33.60	57.40	6.70	1.50	13,860			
Brookwood	0.66	29.61	59.68	10.04	1.07	13,671	2,800	2,910	2,730
	3.58	28.74	57.89	9.74	1.00	13,270			
Carter	0.85	30.41	60.53	8.00	0.91	13,977	2,540	2,850	2,730
	2.80	30.00	59.33	7.86	0.90	13,699			
Cedar Cove	1.20	34.30	57.60	6.90	2.40	13,900			
	4.10	33.30	56.00	6.60	2.30	13,500			
Clark	1.15	35.68	55.61	7.55	0.65	13,915	2,060	2,500	2,210
	2.31	35.26	54.96	7.46	0.64	13,750			
Corona	1.28	39.37	49.54	9.81	2.28	13,030	2,230	2,440	2,360
	2.44	38.91	48.95	9.70	2.25	12,878			
Gholson	1.31	35.79	59.36	3.53	0.66	14,480	2,090	2,410	2,210
	3.01	35.17	58.34	3.47	0.67	14,232			
Glass	0.70	31.60	55.50	12.20	0.70	13,280	2,390	2,450	2,420
	2.50	31.00	54.60	11.90	0.70	13,030			
Gould	0.75	30.90	60.32	8.03	1.87	13,934	2,450	2,510	2,470
	2.97	30.21	58.97	7.85	1.83	13,622			
Harkness	1.21	33.27	55.31	10.21	2.02	13,393	2,090	2,660	2,300
	2.33	32.90	54.67	10.09	2.00	13,319			
Helena	1.23	34.43	55.76	8.57	0.62	13,569	1,910	2,790	2,270
	3.71	33.53	54.36	8.38	0.59	13,255			

Henry Ellen	1.43	33.50	53.50	11.57	0.88	13,212			
	2.52	33.13	52.91	11.44	0.87	13,091			
Jagger	1.48	29.72	52.92	15.87	0.95	12,336	2,390	2,930	2,670
	2.95	29.29	52.09	15.66	0.94	12,213			
Jefferson	0.81	33.22	59.34	6.61	3.22	14,151	2,060	2,420	2,260
	2.25	32.74	58.48	6.52	3.15	13,947			
Mary Lee	1.11	30.37	56.08	12.42	1.11	13,072	2,200	2,950	2,740
	2.86	29.82	55.11	12.19	1.10	12,841			
Maylene	1.40	37.15	53.95	7.50	0.45	13,603	2,190	2,470	2,330
	2.99	36.55	53.08	7.38	0.44	13,070			
Milldale	0.61	31.75	63.03	4.60	0.93	14,566	2,090	2,800	2,420
	2.11	31.26	62.09	4.52	0.91	14,347			
Montevallo	1.36	37.21	53.85	7.57	0.84	13,719	2,090	2,470	2,330
	2.51	36.78	53.22	7.48	0.83	13,481			
New Castle	1.00	31.08	53.89	14.03	2.04	12,849	2,390	2,510	2,450
	2.97	30.46	52.82	13.75	2.00	12,593			
Nunnally	0.63	34.97	56.27	8.13	0.97	13,787	2,190	2,410	2,300
	2.22	34.41	55.37	8.00	0.95	13,566			
Pratt	0.51	29.66	62.81	7.00	1.59	14,317	2,180	2,850	2,390
	2.45	29.60	61.46	6.38	1.57	14,149			
Thompson	1.79	35.73	55.53	6.93	0.60	13,646	2,060	2,400	2,190
	3.00	35.29	54.83	6.86	0.59	13,518			
Woodstock	1.47	35.22	59.57	3.73	1.07	14,390	2,110	2,390	2,240
	3.19	34.61	58.52	3.67	1.05	14,138			
Yessick	2.35	30.43	46.24	20.98	1.57	11,221	2,240	2,340	2,280
	4.21	29.85	45.36	20.58	1.54	11,008			
Youngblood	1.08	35.25	58.39	5.28	1.28	14,395	2,010	2,220	2,090
	2.71	34.67	57.43	5.19	1.26	14,373			
Mount Carmel	1.41	30.52	54.85	13.17	0.81	12,840			
	3.14	29.98	53.90	12.96	0.80	12,506			

* Table compiled by J. J. Forbes, Bureau of Mines, Birmingham, Ala., analyses made at the Laboratory of Bureau of Mines, Pittsburgh, Pa.

(a) Analyses on "air dried" basis; this order applies in each seam given.

(b) Analyses on "as received" basis; this order applies in each seam given.

horizon, not regularity as to thickness, quality, or pitch. The seams vary in thickness from a few inches to 16 ft. All the seams are more or less variable and none is of workable thickness for the whole length of its outcrop nor as the seams extend into the field.

The Cahaba field lies southeast of the Warrior field, from which it is separated by the Silurian and Cambrian measures. This field is 68

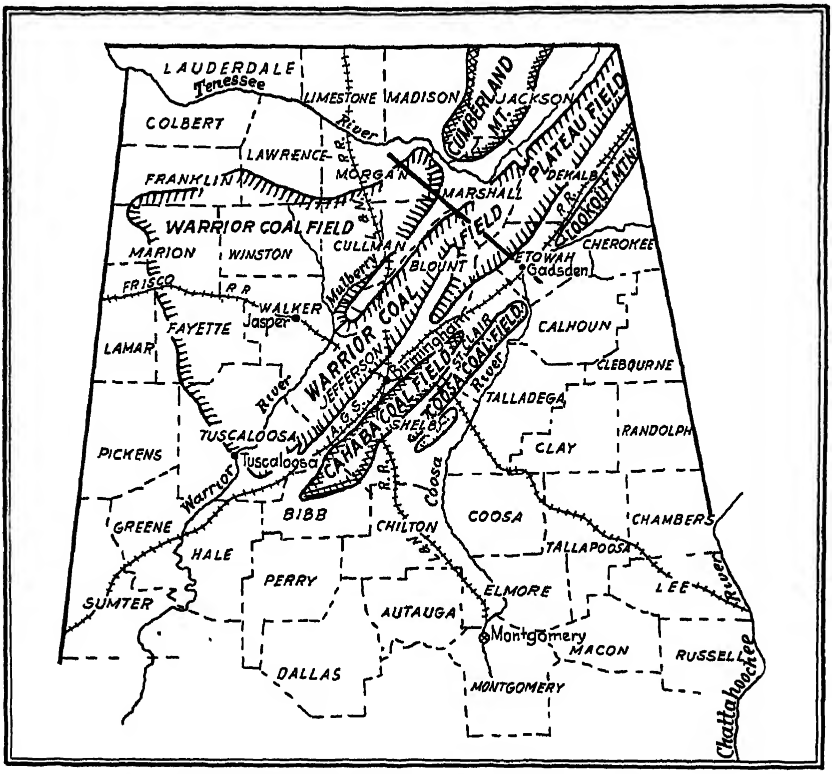


FIG. 2.—MAP OF ALABAMA COAL FIELDS.

miles long and from 5 to 8 miles wide; it contains a surface area of 394½ sq. mi. There are eleven basins, not including the "Overturned Measures."

Coal seams in this field are principally pitching. The roof conditions generally are not so good as in the Warrior field, and more pumping must be done. Considerable methane is encountered, which together with a variation in the pitch and thickness of the seam, makes mining more difficult than in the Warrior field. Columnar sections of the coal strata in both the Warrior and the Cahaba fields are shown in Fig. 3.

The Coosa field lies mainly in Shelby and St. Clair Counties and contains an estimated area of 340 sq. mi. The production in this field may be considered as negligible; in 1923, no tonnage was produced.

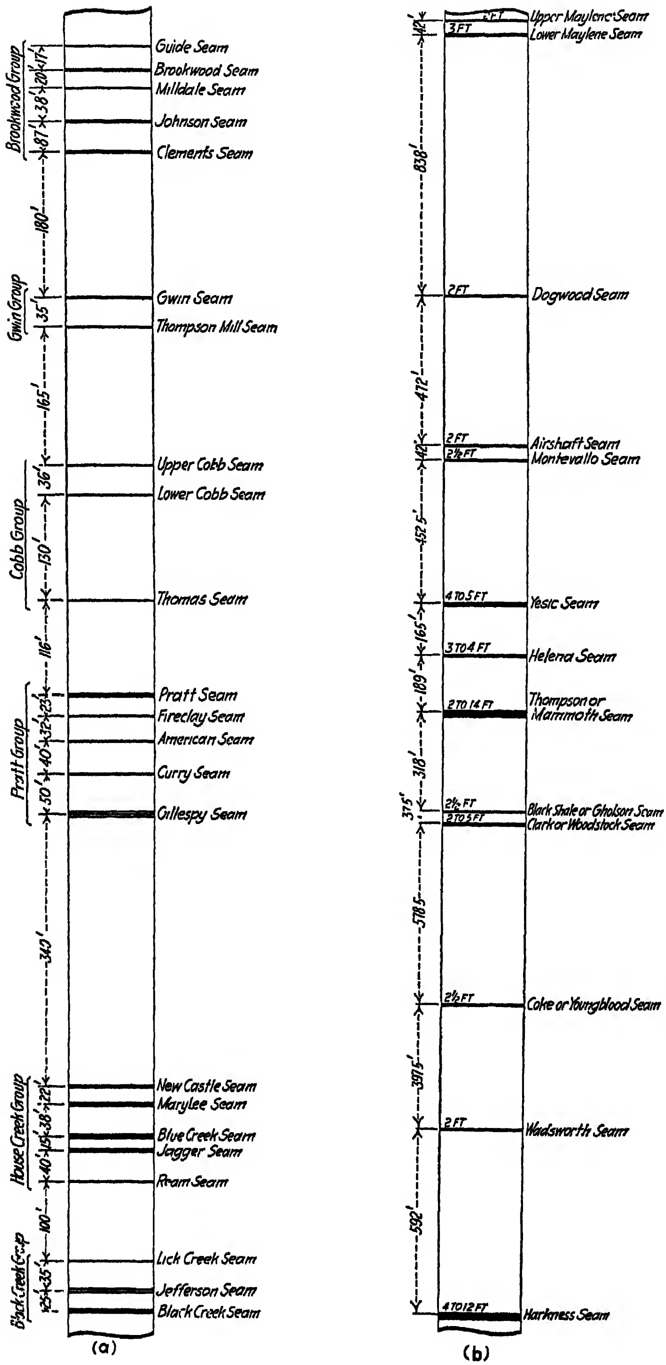


FIG. 3.—(*a*) SECTION IN SOUTHEAST EDGE OF WARRIOR COAL BASIN, IN BROOKWOOD DISTRICT; (*b*) SECTION IN CAHABA COAL BASIN, NEAR MONTEVALLO.

The Plateau region includes those coal measures known geologically as the Plateau field. The Plateau region consists "of the high, wide, flat and plainlike areas of the tops of the Cumberland Mountain, Sand Mountain, Raccoon Mountain, and Lookout Mountain." This region is better described as being that area of the coal fields of the state running from Attalla north and east. The Plateau region, of which the coal measures are a part, comprises about 4500 sq. mi. The coal measures in this field have never been mined profitably. Several furnaces north and east of Birmingham were built in this region, with the hope of procuring an adequate and suitable fuel and ore supply; but all of these ventures proved failures, as far as procuring raw material in this region. The coal seams in this field are pockety, irregular, and uncertain.

Examples of Irregularities in Coal Seams

When the term "regular" is applied to coal seams in Alabama, it is used only in its geological sense and not with the meaning given by

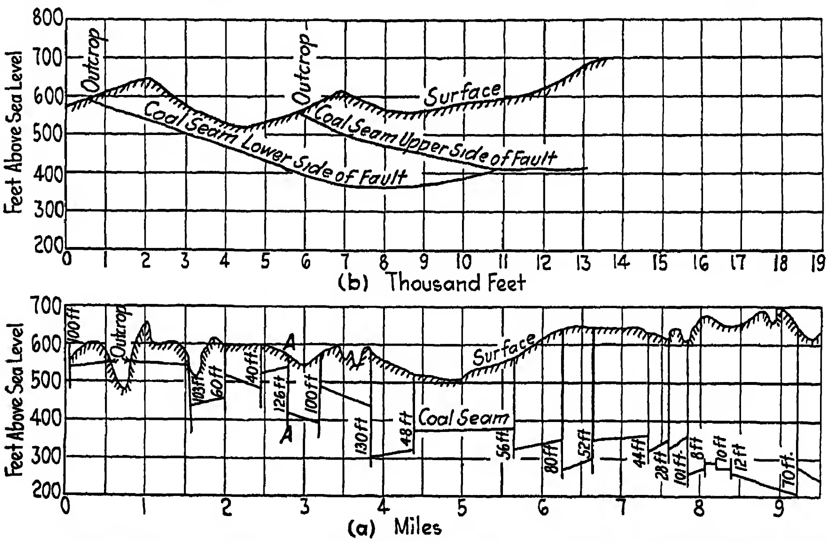


Fig. 4.—Profile parallel to and ½ mile from outcrop, showing dislocation faults proved in mines.

the engineer. In the Warrior and Cahaba fields, the chief producers, the pitch, thickness, and character of the seams vary greatly in a limited area. While the examples here cited probably should not be considered typical, they are fairly illustrative of that with which the producers of coal in Alabama must contend.

In the Warrior field, adjacent to Ensley, the greatest quantity, as compared to all the other seams, of high-grade coking coal has been produced from the Pratt seam. Some idea of the difficulty of coal mining in Alabama may be gleaned from Fig. 4 (*a*) which shows a profile

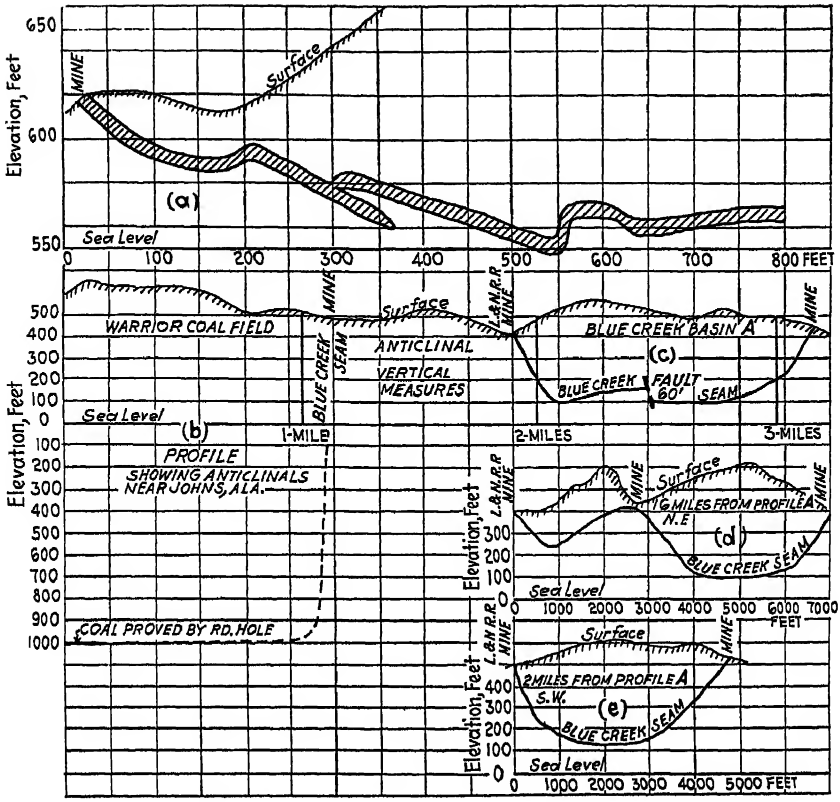


FIG. 5.—PROFILE IN BLUE CREEK SEAM ALONG LINE OF APACHE SLOPE, SHOWING THRUST FAULTS.

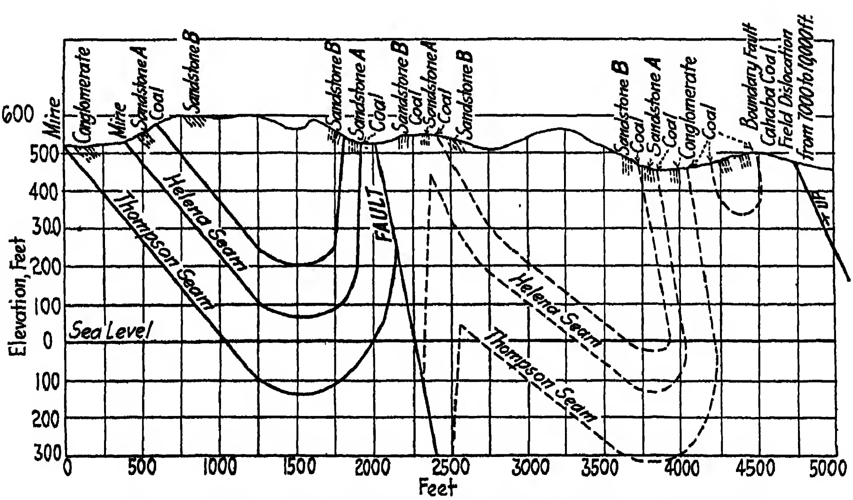


FIG. 6.—PROFILE ACROSS HELENA BASIN, CAHABA COALFIELD.

parallel to, and ½ mile from, the outcrop, showing dislocation faults actually proved by mine workings on the Pratt seam in the Warrior field. In 9 miles there are nineteen faults, the throw of which varies from 8 to 130 ft., and there is no regularity in the pitch of the coal. In (*b*) is shown a thrust fault along *A-A*.

Some of the problems of Alabama coal seams are illustrated in Fig. 5. A profile of the Blue Creek basin of the Warrior field, showing thrust faults, as proved from mine workings is shown at (*a*); a profile of the anticlinials near Johns, in the same basin is shown at (*b*). At (*c*) and (*d*) are profiles through the same basin, (*c*) being 1.6 miles northeast of (*a*) and (*d*) 2 miles southwest. A profile across the Helena basin in the Cahaba field is shown in Fig. 6. The full lines represent the seams as proved from mine workings to the fault; the broken lines are determinations of the continuation of these seams from geological studies.

Mining Methods

Coal is won from thirty workable coal beds that range in thickness from 2 to 13 ft., with an average thickness of 4 ft. The principal producing beds in the Warrior field are flat, or nearly so, while those of the Cahaba range from slightly to comparatively heavy pitching beds. It is to be expected that, on seams from 2 to 13 ft. thick, pitching from 0° to 90°, nearly all mining methods are used; hence a description of such methods must be more or less general.

Mines are opened, generally, through drifts and slopes, although there are some vertical shafts in the state. The principal method of mining is the ordinary room-and-pillar. A modified longwall system is being used in several mines in the Cahaba field on pitches, and modified longwalls, or panels, are being used in several mines of the Warrior field. Some of the coal beds have fairly well-defined cleats while others are not so well defined; consequently, little attention is given to mining on the butts and face cleats of the coal, with the exception of those having defined cleavage planes.

In order to classify seams as to mining methods, the following is adopted: Class I, flat seams, or nearly so; Class II, seams pitching from flat to 22°; Class III, steeply pitching seams, over 22°. All of these classes should be subdivided as to thickness, as follows: (*a*) Thin seams, 36 in. and under; (*b*) medium thickness seams, from 36 in. to 4½ ft.; (*c*) thick seams, over 4½ feet.

Description of Mining Methods

Thin Flat Seams

To mine seams under 36 in. successfully the conditions must be favorable; most of the thin seams mined lie flat, or nearly so. It has not

been found profitable, with very few exceptions, to mine thin seams that are medium or steeply pitching. In most instances, thin seams lying flat, or nearly so, are mined through drifts opened on the coal, although in a few cases such seams are reached by means of short slopes or shallow shafts.

Where this type seam is mined through drifts, the coal outcrops in hollows, and tram roads are graded along these hollows. Drifts or entries are turned into the hills, usually at room distances apart, either with double entries or an air course with each entry; and where the dip does not prevent, rooms are turned from both entries, where the double-entry system is used, or from the entry and the air course. Where pitch does not permit, rooms are turned only from headings and the air course serves for ventilation. In some cases, room necks are connected by crosscuts and closed with brattice cloth, these crosscuts serving as an air course. This practice applies only to drifts of short life. The depth of the rooms in thin coal lying flat varies from 125 ft. in 26 or 28-in. seams to 175 ft. in 36-in. seams. The empties for the miner are delivered on the entry, and he delivers the loads there. The thickness of pillars varies from 10 ft., with rooms from 20 to 30 ft. wide, to 40 ft. with 40-ft. rooms. The thickness of cover varies from 40 ft., which as a rule is considered a minimum and then only in places, to 300 or 400 ft.

Where a large body of coal is accessible to drift mining, one main pierces the property, and cross entries, at regular intervals, are turned from the main entries. The mode of obtaining coal in these entries is the same as where the entries are turned from the surface. The method of mining in the past, through a drift, has generally been inefficient from the standpoint of recovery and ventilation, but in more recent years more effective systems have been applied.

A panel system in use in a drift mine in thin coal is shown in Fig. 7. The main entry *A* is opened from the surface and the main face entries *B* and *C* are parallel to and 1600 ft. from it. Main butt headings, from which no rooms are turned, connect *A*, *B*, and *C*. Double entries *D* and *E*, five to a panel, are turned from the main and main face entries. Each panel is surrounded by a barrier pillar that is ample to protect it. The headings in each panel *D* and *E* are developed by double gob entries, as shown in Fig. 8. Rooms are turned on 80-ft. centers, each room having a double neck and with a double track. After the room is driven in from 50 to 65 ft., it is widened to connect with an adjoining room, until five rooms are connected. This affords a 400-ft. face, which in some instances has been carried forward for 70 ft., although this face usually advances in steps. To afford the miner an opportunity to load, switches are laid from the room track or, in some instances where the top permits, the track is curved around the face. A solid pillar 40 by 145 ft. is left after five rooms are turned; when expedient, these pillars are left, so that

on one entry the pillar comes beyond room No. 5 and in the next entry beyond room No. 7. The purpose of this panel system, in thin coal lying flat, is to afford means for robbing each entry as it is worked to its limits by leaving pillars of sufficient size to obtain a maximum recovery. The recovery of pillars in thin coal is difficult at best, and it has been found more satisfactory to leave short thick stumps rather than long thin pillars, which had been the practice in this state until recently. Of course, this method can be applied only where the ownership is in fee. Where the ownership does not include the surface, rooms 30 to 40 ft. wide, depending on the top, are turned with pillars from 10 to 20 ft. in width, depending on the depth of the overlying strata.

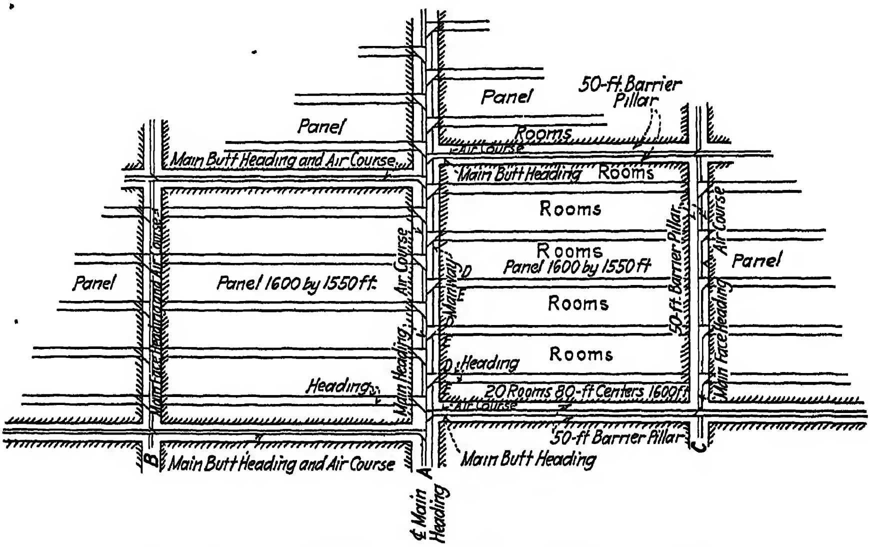


FIG. 7.—PANEL SYSTEM IN DRIFT MINE IN THIN COAL.

In the gob entries, which are driven usually 5 ft. 2 in. above the rail, the rock above the coal, varying from 36 to 44 in., is shot down and gobbed. The entries are driven in the coal of sufficient width to provide room for this rock. In coal 30 in. thick with entries 5 ft. 2 in. above the rail, the entries must be driven 28 ft. wide in the coal to take care of the rock in an entry of standard width; namely, 9 ft. wide at the bottom and 7½ ft. wide at the top. The cost of such yardage varies from $2.56 to $4.50 per running yard, depending on the thickness and hardness of the rock; the cost of yardage per ton varies from 25 to 40 cents.

Where the hardness of the rock necessitates, jackhammer drills, with portable air compressors, are used for drilling rock holes. Where such drills are used, it is common for the miner to drive his entry into the coal, say from 60 to 100 ft. ahead of his brushing. He then comes back and sets his breaking timbers, after which the drill runner spends the entire

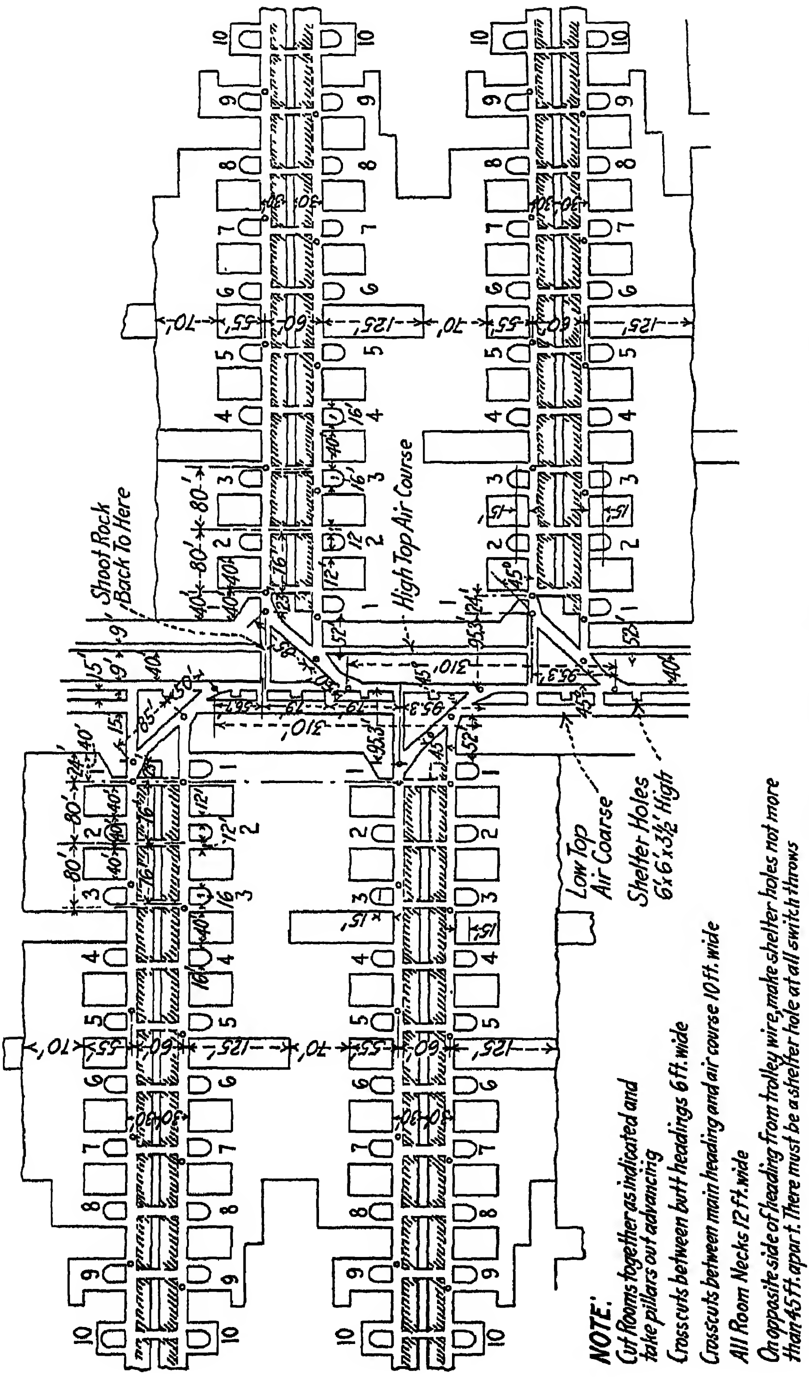


FIG. 8.—DEVELOPMENT OF PANELS IN A DRIFT MINE IN THIN COAL.

shift in the entry drilling the rock. A disadvantage of this method is that the miner cannot load coal each day, and utilize his spare time for handling his rock, but it enables the drill runner to accomplish better results for he does not have to transfer his drill and compressor during the shift. Coal lying as described is usually undercut with electric machines of the low-vein type. In some instances, one track of the room is brushed, that is, from 6 to 18 in. of rock is taken down over the roadway to admit the machine. There are some modifications of the method described of mining thin coal lying flat. At one mine, in 30-in. coal, an experiment is being made with mechanical loading. While this method bids fair to prove successful, its consummation is not definitely assured.

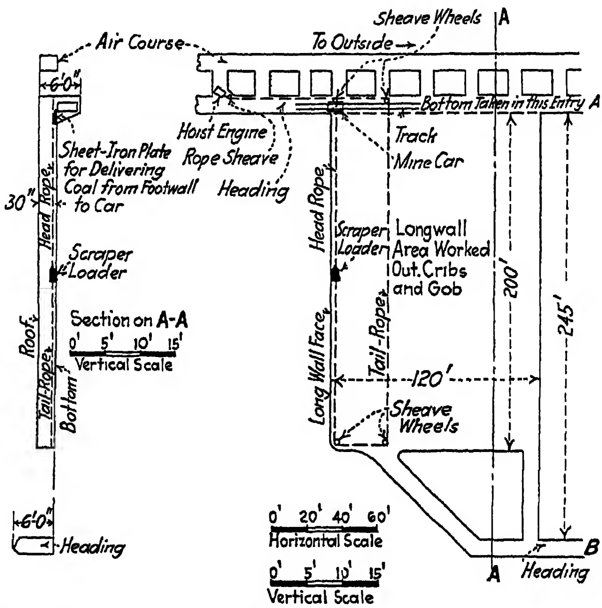


FIG. 9.—ENTRIES IN A DRIFT MINED SEMI-LONGWALL, MECHANICAL LOADING.

Two cross entries, as indicated in Fig. 9, were turned in a drift mine 245 ft. apart. A pillar 75 ft. wide was left to protect the main entry and the cross entries were connected. In one cross entry *A* the bottom was taken up instead of brushing the top; in the other entry *B*, the top was brushed to entry height. The bottom consists of fireclay 3 to 4 ft. thick, with a hard shale about ½ in. thick directly under the coal. Below the clay is sandstone. Above the coal is from 5 to 15 ft. of siliceous shale, and above this shale a hard, impervious sandstone 20 ft. thick; above this sandstone are shale and sandstone. The entire thickness of the cover over the coal varies from 75 to 150 feet.

The face, or wall, 200 ft. in length, is mined advancing. The coal is undercut with a chain machine, with 6-ft. cutter bar. Four rows of

heavy timbers, set about 4 ft. apart, were placed up to 5 ft. of the face, and as the wall advances, the rear timbers are removed. A drag loader, 4 ft. wide of the Goodman type, loads the coal directly into the cars placed in entry *A*, which is driven 250 ft. ahead of the wall. Cribs placed "skin to skin" in two rows protect both entries. When the wall had advanced 100 ft., a break occurred and the fall rode the face to a slight extent. The wall was cleared and, instead of using timbers along the face, cribs were built every 10 ft. and left in place, with a result that the overburden crushed the cribs located back from the face without any falls occurring at the face. There has been no indication of the fireclay bottom heaving to the extent of disturbing the entry *A*.

From an application of this system an apparent saving of 25 to 50 cents per ton has been effected, when the work progressed smoothly. In addition, the lump percentage was increased. The problem is one of control of roof subsidence, which must be worked out through experience, and the training of labor who have no experience on walls.

Mining a Flat Seam of Medium Thickness

Much of the coal used in the manufacture of coke for the blast furnaces of the state is mined from a flat seam of medium thickness, the Pratt. In the smaller mines, the prevailing method of mining is to drive a main opening through the center of the area to be worked. This opening is driven for single or double track, depending on the tonnage desired. It is necessary to blast down about 18 to 24 in. of roof to give height for a good haulageway. In some mines, a slab is driven with the haulage way wide enough to gob all the brushing rock; in other mines, the brushing rock is hauled to the surface, or gobbed in crosscuts, and rock eyes cut at intervals along the ribs.

An airway 12 ft. is driven on each side of the main haulageway, a pillar 30 ft. thick being maintained. Room entries are turned from the main haulageway approximately 300 ft. center to center which, after allowing 18 ft. for the width of the entry, including the slab, 20 ft. for the airway pillar, and 12 ft. for the width of the airway (a total of 50 ft.) there remains 250 ft. for the length of the room. This is the average depth of rooms worked in the Pratt seam. Some mines have worked rooms 300 ft. deep, but as the miner receives his empty car on the entry and has to deliver the loaded car on the entry, the tendency has been to decrease rather than increase the length of rooms. The lay of the seam is watched closely and, as far as practicable, all rooms are driven to the rise, thus keeping the room dry and giving the miners, who must push their cars, a grade in favor of the loads.

In the larger mines, the area to be mined is unusually carefully blocked out and worked on the panel system. Butt headings, or haulage-

ways, are driven at right angles from the main haulageways and at intervals of from 1600 to 2000 ft. and room entries are driven from these haulageways at intervals of 300 ft. These panels, or sections, are surrounded by barrier pillars, ranging from 100 to 200 ft. thick, depending on the amount of cover over the area to be mined.

With one or two exceptions, all the mines on the Pratt seam are using the room-and-pillar system. The width of the rooms and pillars vary, depending on the character of the roof and the thickness of the overburden. The most popular system is a single-neck room driven 35 ft. wide and 250 ft. deep. The rooms are turned on 70-ft. centers, which provides for a 35-ft. pillar. A track is carried on each rib of the room so as to reduce the amount of turning of the coal, and makes the loading of the coal into the car convenient for the miner.

Owing to danger of encountering "squeezes," the room entries are driven to the boundary and all rooms worked out before any attempt is made to recover the pillars. The pillar robbing is started in the next to the last room—a track being carried up the rib on the inby side of the room—and is carried through the last cross-cut, which is at the face of the room. The coal is sliced off of the end of the pillar by machines or hand mining (in most instances pillar work is done by pick miners) and the track is moved as necessary over against the coal, so as to provide easy loading and permit the setting of timbers. After the first pillar has been pulled back about 50 ft., a second pillar is started; in the same order the third and fourth pillars are started. An attempt is made to keep the face of the pillars on a line running at 45° with the room entry. The room pillars are pulled back to within 30 ft. of the entry and are left standing for a short time until they are extracted by miners known as "stump pullers."

Where the roof conditions will permit, double-neck rooms are turned and are driven 50 ft. wide with a 50-ft. pillar between rooms. This width of room is popular with the miners, and also provides a large tonnage per mining machine. The recovery of pillars on these 50-ft. rooms has been exceptionally high; the pillars are so thick that they are easily recoverable and, in cases where the rooms on each side of the pillar have fallen in, the pillar being 50 ft. thick, can be slabbed up or split, and practically all of the coal recovered. In the best regulated mines, where all rooms are driven on centers and the pillars are maintained according to some well-established plan, the recovery of coal runs as high as 95 per cent. The top, generally, is good; in some instances, however, local spots of bad top are encountered. The lay of the seam is fairly uniform and rooms can usually be driven so as to go to the rise; but where local swags cause the rooms to go to the dip the rooms are either abandoned and driven back from the heading beyond, or the cars are hauled from the rooms by room hoists.

Several mines in the district are now experimenting with semi-long-wall mining, the work being done in conjunction with mechanical loaders. As a matter of fact, it is really two experiments being handled as one. All the work is still in an experimental stage and so far no definite conclusions have been reached.

At one mine, which so far has obtained the best results in the district, two entries 500 ft. apart were driven to the boundary, and then the face was opened up by driving a room 20 ft. wide from each side, these rooms meeting at the center of the block of coal. One cutting machine equipped to take a cut 4 ft. deep undercuts the place on the night shift. The coal is next blasted down, the length of the wall timbered, and left ready for the day shift, or the loading crews. Two loading crews are used, one working from each entry and being assigned to 250 ft. of the wall. The entries were driven 9 ft. wide and 4 ft. of the bottom was lifted so as to place the mine car below the level of the coal seam.

When the work was begun, sectional conveyors were used for loading, but were abandoned and two drag loaders, one for each entry, were installed. Cribs, as previously described, are used to protect the entries and to protect the walls. The cribs protecting the walls, placed 14 ft. lengthwise of the walls and 8 ft. apart, are not moved but are left in place as the wall retreats. The cover over these mines varies from 150 to 500 ft. and is composed of sandstone and shale. The bottom is a hard sandstone.

The actual loading is performed by what is known as the drag system. A double-drum electrically driven hoist is placed on the entry directly opposite the cut of coal to be loaded; a scoop, or drag, is placed on the face and a rope fastened to each end of it. The rope fastened to the rear end of the drag is carried to the rear end of the section and passed around a sheave, thence to what is known as the tail-rope drum. By operating the hoist, the drag is hauled along on the bottom, backward and forward, the length of the face and is in position to transfer the coal from the face to the mine car, which is located on the entry opposite the cut of coal.

The drag is made in the form of a "V"; its sides are approximately 18 in. high, its length is usually 5 ft., the opening at the front is 4 ft., and it has a carrying capacity of 1500 lb. of loose coal. It is made to "bite in" to the loose coal by arranging the sheaves so that the rope is held against the face of the coal. A sheet-iron chute or pan is placed on the entry, which overhangs the side of the car to be loaded and provides a cheap practical way of transferring the coal from the drag to the mine cars. Experience with mechanical loading, generally, is more or less a matter of conjecture; its success has not been proved. On account of the partings in the thicker seams, mechanical loaders of the Joy type have not been used.

Mining Flat Thick Seams

Thick seams lying flat are mined in practically the same manner as thin seams and those of medium thickness. The rooms are deeper and the development necessary for a given tonnage is less; the production cost in some instances is only one-half the cost of producing coal in seams 30 in. and under. Under favorable conditions, seams 5 to 10 ft. thick, with partings from 24 to 30 in. are profitably mined. In some cases the miner actually removes and gobs ½ ton of rock for each ton of coal loaded. When such are mined with a chain machine, the bottom bench is shot and loaded, the "middleman" is then forced down with a bar or wedge and gobbed and the top coal is then shot and loaded. A section of a seam with a 12½-in. "middleman" so mined is shown in Fig. 10. The washer loss at this mine is in excess of 20 per cent.

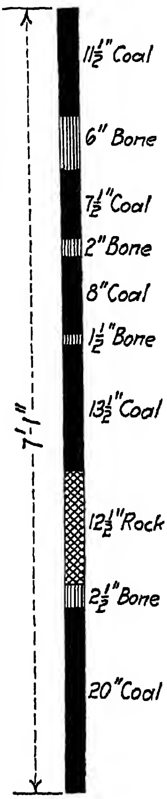


Fig. 10.

Rooms in a mine on one of the thick seams, lying comparatively flat, are shown in Figs. 11 and 12. Fig. 11 shows that one-half of a room's width is driven ahead of the other half, the coal shot down and loaded while the other half of the room is being undercut by hand. This is a safe method, for the miner is required to leave a heel, or knee, on the loose end and in the center of the undermined half of the face. These knees are cut out before the shots are fired. After the miner has undercut the coal as shown, he must stand up while he cuts out the heel, so that the hazard from falling coal is minimized. A safety travel way for the miner is provided in all rooms; as shown in Fig. 12, it parallels the room track.

Strip Mining

There are some eight or ten coal-stripping operations in Alabama, all are on seams that lie flat; coal from 28 in. to 6 or 7 ft. is being mined. From 4 to 5 ft. to 30 and 40 ft. of cover is being removed from the coal with shovels of the larger type; the small, or loading, shovel follows the stripping shovel and loads the coal either into railroad or mine cars. Practically all the stripping coal is washed before it is shipped to the consumer.

A successful operation is shown in Fig. 13. At this operation levees are built and the creek changed when necessary to protect the open cut. The coal, 42 in. thick with a 4-in. parting, is stripped with a 150-B Bucyrus revolving shovel, regularly equipped with a 2½-yd. dipper and a 60-ft. boom. The base is of rigid construction and the trucks are of the equalizing type.

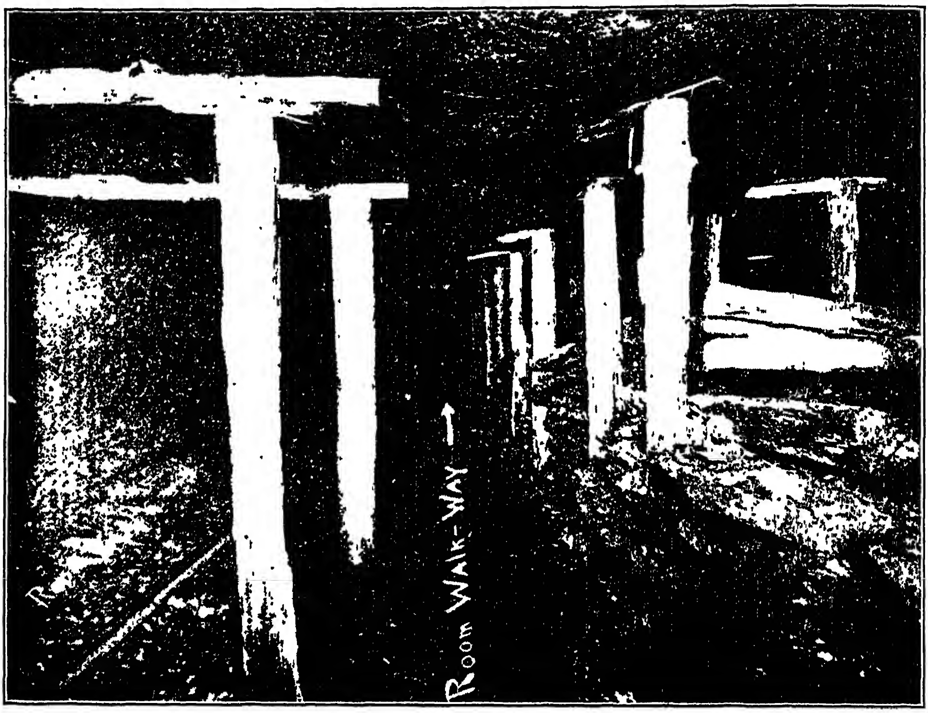
This shovel can make a box cut in overburden up to 18 ft. in depth, depositing the excavated material on one side. After the box cut has been completed, its operating range permits the shovel to range in overburden up to 25 or 28 ft. in depth. This shovel has progressed at the rate of 90 ft. per 24 hr. in a box cut 55 by 18 ft. or 142 cu. yd. per hr. The output of this shovel depends on conditions, but a fair average, allowing for moving up and other delays, is 150 to 200 cu. yd. per hour.

(*Republic Iron & Steel Co.*)

FIG. 11.—ROOMS IN A THICK COMPARATIVELY FLAT SEAM.

After the coal is stripped, it is loaded with a 35-B Bucyrus standard revolving shovel with a few changes over the standard design. It carries a longer boom and a dipper handle, 30 and 19 ft., respectively, and a special flat-front coal-loading dipper. The former enables it to cut the wide swath made necessary by the great range of the coal-stripping shovel that has preceded it, and gives it the higher lift necessary to load coal into a gondola, or other type of car upon a 6-ft. bank. The dipper has a capacity of 1½ yd. It is built with a flat front, which enables it to dig the coal without undue breakage.. It is regularly mounted on caterpillar tractors, so that it may be moved readily and eliminates the use of platforms or tracks, with the extra pit labor. The coal is loaded directly into the railroad cars. Cars are brought, one at a time, by a

small locomotive, from the storage track, a haulage of 2000 ft. A car of 55 tons capacity is loaded in 13 min.; but because of the time required



(*Republic Iron & Steel Co.*)

Fig. 12.—Safety travel way for miner.

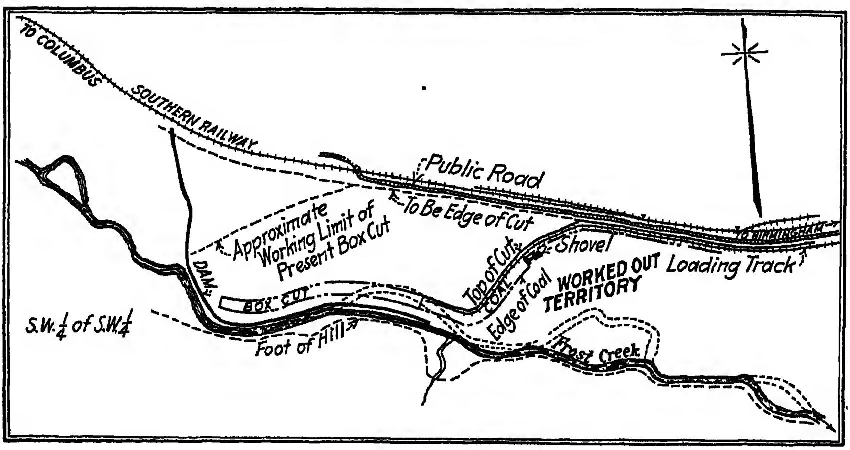


Fig. 13.—Successful strip mining operation.

to place the cars, the shovel will load on the average one railroad car per 30 min.

At this mine, the total overburden is prepared for the stripping shovel by drilling holes, on 18-ft. centers, to within 1 ft. of the coal and shooting them with two and one-half sticks of 40 per cent. gelatin, using a 5 by 16 in. cartridge. The box cut is approximately 1100 ft. long, and develops approximately 14 acres or 90,395 tons. The box cut corresponds to narrow work in underground workings. After the box cut is driven, slabs are removed along the face until the workings advance to their limits, when another box cut is driven into the property.

There are some variations in this method. At some operations, the shovels work in a circle; in others, they work along the side of hills until the overburden becomes excessive. Where conditions are favorable to strip mining, it has been found that the cost of labor per ton for coal delivered on the railroad cars is from 50 to 70 per cent. of the cost of underground mining for a seam of like thickness.

Medium Pitching Seams

Where a thin seam was treated in detail, as an example of a flat seam, a medium thickness seam will be considered typical of the medium pitching class; seams of this class are found principally in the Cahaba field.

Conditions in the Cahaba field are extremely variable. All the seams are pitching; in seams now being worked the dip varies from 5° to 49°. The roof over the coal varies from fairly good to bad; in some instances, entries and roadways in rooms are cross-collared. The rooms are timbered closely with heavy capboards over props. The bottom varies from hard to soft. With these extremes in mind, it is manifest that systems of mining and timbering are different in different mines, and even in the same mine.

The coal seams in this field are opened on slopes that follow the coal from the outcrop (see Fig. 14). At intervals of from 200 to 300 ft., depending on the thickness of the seam, entries or lifts are turned right and left on the strike from the slope. The slopes have one or two air courses paralleling them, with a pillar 25 to 50 ft. between. When two air courses are driven, one is brushed, if the thickness of the coal makes it necessary, to a height of 6 ft. for a manway; this manway is usually 10 ft. wide. The other air course is from 6 to 13 ft. wide and the height of the coal. Air courses parallel the cross entries, generally on the dip side. The entries are driven "water level;" that is, following the undulations of the seams on slightly rising grades, to facilitate transportation and drainage. Rooms are turned off the lifts to the rise on 35 to 60-ft. centers and coal is won, where the pitch is not too steep and the thickness of the coal permits, by delivering the car at the face with mules or by the gravity method.

The general practice is to use the gravity method on seams of this class, where the pitch varies from 5° to 20°. Rooms are driven to the

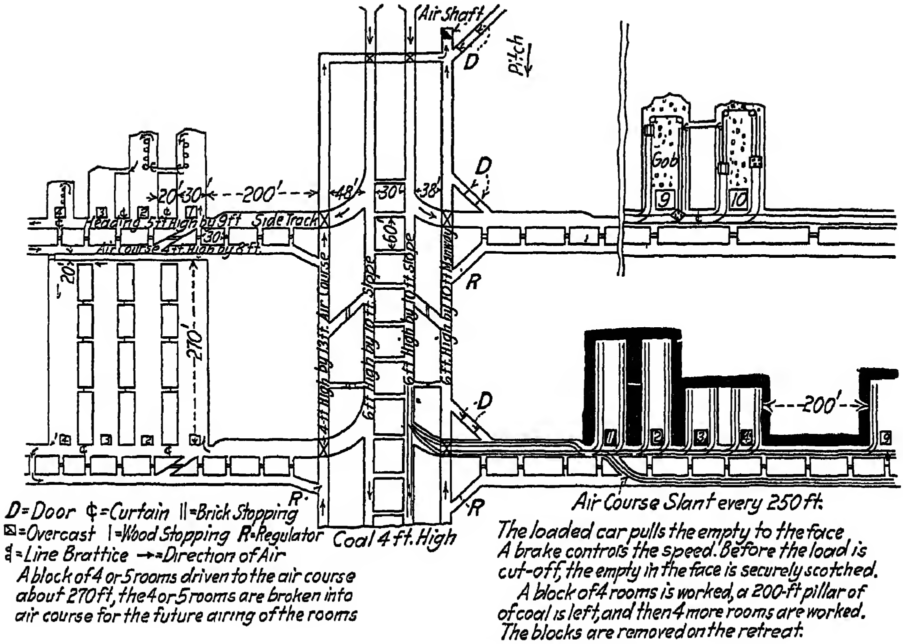


FIG. 14.—MINING MEDIUM PITCHING SEAMS.

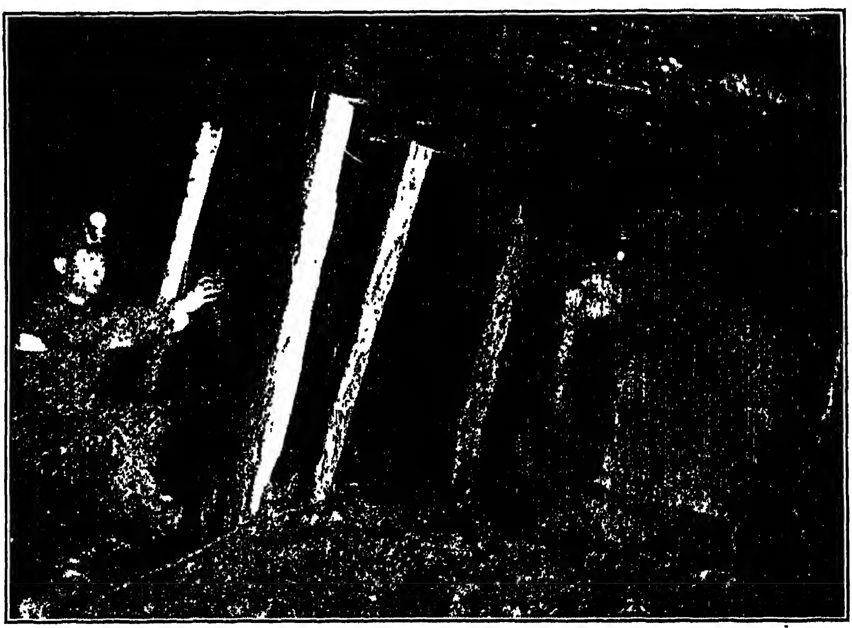


FIG. 15.—METHOD OF SPRAGGING INCOMING EMPTIES AND OUTGOING LOADS, GRAVITY SYSTEM, CAHABA COAL FIELDS.

rise and double tracked, with track running close to ribs and extended to about 10 ft. of the room face. Pulleys, from 8 to 12 in. in diameter, are clevised to posts set about 5 ft. in advance of the room tracks. A rope of ample length to care for room advancement, encircling the pulleys, is attached to the loaded and empty cars. An improvised braking device, see Fig. 15, situated midway between and in line with pulleys, enables the miner to regulate the speed of the outgoing car, pulling the incoming empty car to the face. Such method requires great care in the laying of room tracks, in the setting of pulley timbers, and in the use of the rope.

In advancing rooms three lengths of rope are used; these ropes are 150, 250, and 350 ft. long. There are two links in each rope, which are fastened to it by clamps, the surplus rope being thrown on the mine cars; these links are moved on the rope as the room advances. When the room has progressed beyond the point where one length of rope can be used, this rope is moved to some other room and a longer length rope substituted. The ropes are 3/8 and 1/2 in. in diameter, depending on the pitch.

It has been found that patent brake pulleys are not successful. Sprags are used in the wheels of loaded cars where the pitch is 12° or over. On this pitch wooden room rails are generally used as the damp steel rail does not give enough friction, even though the wheels be spragged, on pitches above 12°.

A modification of mining a seam of medium pitch is shown in Fig. 16. The mine is developed through a slope, together with an air course and a manway. Level or strike headings driven double, spaced 700 to 800 ft. apart and averaging about 10 ft. wide, are turned right and left off the slope. Auxiliary slopes, or "dips," are turned off headings about every 600 ft. The system of mining employed is the room-and-pillar panel with all wide work paralleling headings and auxiliary slopes paralleling the main slope. Hoist headings are driven to the rise a distance of 100 to 120 ft. off headings for the purpose of handling coal from dip panels. Rooms, averaging 24 ft. wide with intervening pillars of 20 ft., are turned right and left off dip slopes and a barrier pillar approximately 40 ft. thick is maintained between the strike entry and the first room turned off dip slopes. Coal is hauled, by rope, out of dip slopes by an electric hoist placed directly in line with the dip slope but to the rise from the heading. From main partings in headings, coal is hauled to a side track, thence up the main hoisting slope to the surface. Coal is undercut with machines in rooms off dips but solid shooting is used for breaking down coal in all narrow work.

There are some modifications of this system, which consists chiefly of a semi-longwall. The application of this system is the same as that later described under thick seams steeply pitching. Where thin and

thick seams are mined on medium pitches, the method used is the same as that described for medium thickness except as to the handling of rock in the thin seams and the depth of rooms in the thin and thick coals.

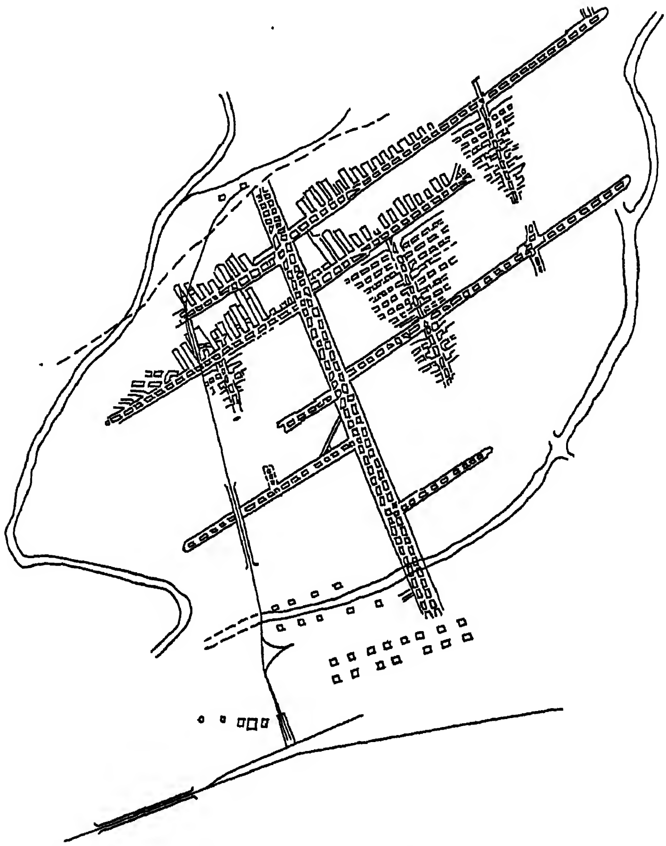


FIG. 16.—A METHOD FOR WORKING SEAMS OF THE SECOND CLASS.

Description of Longwall Method

At one mine in the Cahaba field, a longwall system of mining has been practiced since 1906, on the Montevallo seam, a typical section of which is shown in Fig. 1. (This seam is of medium thickness and medium pitching.) This mine, Fig. 17, was first developed through a slope on a room-and-pillar system, the rooms being cut together after they were necked and driven up the pitch. This system was continued until the slope had advanced about 2400 ft. when the mine was changed to the longwall system. After the slope was driven through the basin, which was only 200 ft. wide, the seam began to rise on about a 12° pitch, and the walls were advanced on the rise. Electric hoists are used to pull the cars up to the pitch. One main hoist raises the empties to a

common point of distribution, and from that point hoists located along the entries leading to the walls pull the cars to the walls and drop back the loads. These hoists are equipped with 27½-hp. motors and have a rope speed of 400 ft. per minute.

The walls are 300 ft. in length and have been mined advancing. Experience has proved that if the walls are kept within 30 to 50 ft. of each other, when the weight comes it is equalized between them. It has also been found that if the walls are kept 250 ft. apart, the falls may be handled on each wall independently, so that when the wall farthest advanced gets a fall, it does not affect the coal through which the adjoining wall must advance. The system of advancing the walls and the location of the hoists for handling the empties are shown in Fig. 17.

When the walls are driven to the rise, a break occurs every 100 to 150 ft. unless a slip or cleavage comes in the roof, which brings on a break more quickly. There is an average of 600 ft. of cover over this mine.

Cribs are placed along roadways and timbers are placed as indicated in Fig. 18. Timbers are used on 4-ft. centers, staggered with 2½ by 4 in. straps; the timbers are 1¼ in. in diameter for each linear foot of length and are left in place. If the gob is inadequate, soft-wood cribs (usually of old timber or soft pine) are built behind in staggered position and left in place to serve as a cushion. The performance and condition of the roof determines whether or not the cribs must be built; where slips or cleavages occur, more cribs are required. If the roof has a tendency to fall more quickly than is normally the case, additional cribs must be built to regulate this feature

The headings are advanced 100 ft. under the coal in the bottom, which consists of alternate coal, rash, and slate. The coal over the heading is then removed as the wall advances. The roadways are cribbed with double rows. The conveyor used along these faces is of the shaker type, an English machine, known as the Mavor & Colson shaker, or Reciprocal Longwall Conveyor; it is made in sections, each 9 ft. long, which are connected by eyelets with bolts. The motor driving the conveyor is placed between the rows of cribs and is connected to the conveyor by a rope drive.

There has been much discussion among engineers as to the proper place for connecting the driving mechanism to this type of shaker. At this mine, experiments to determine the proper place showed that it was more successful to drive the pan on the discharge end. The machine cuts in the rash directly over the bottom coal (see Fig. 1, Montevallo seam). Experience has demonstrated that it is most satisfactory under this system of mining to make the depth of the cutter bar on the machine 1 ft. for every foot of height of coal; in other words, a 4-ft. seam would require a 4-ft. cutter bar.

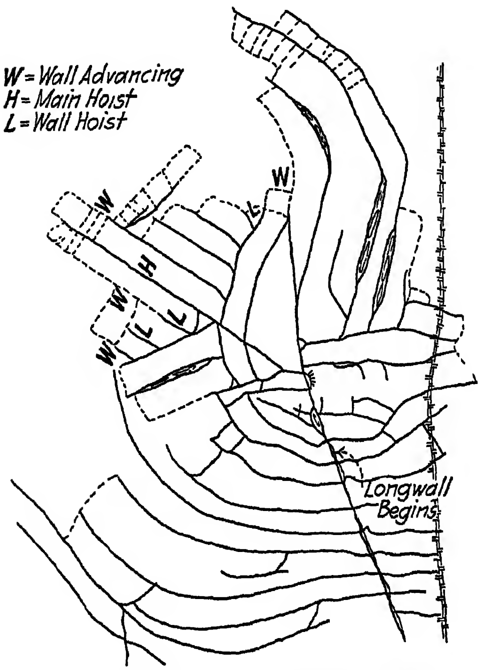


Fig. 17.—Sketch of longwall mine.

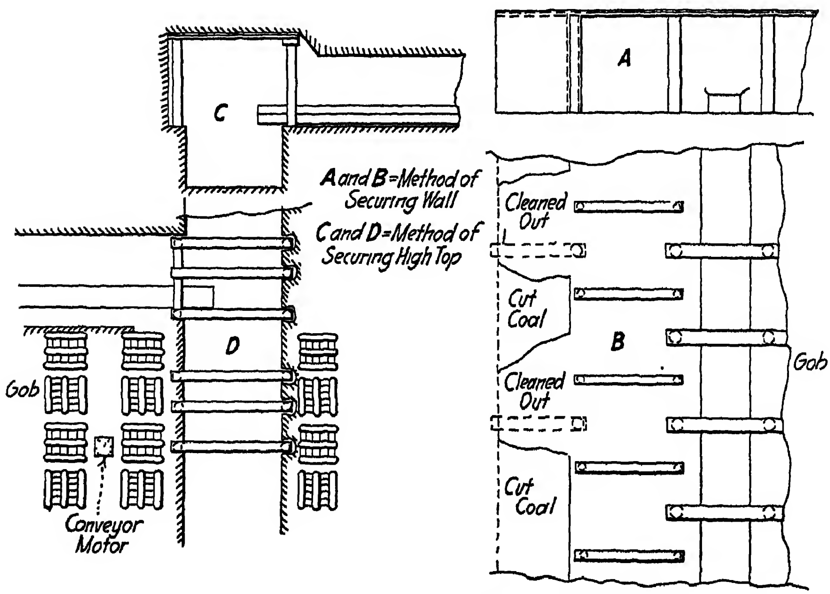


Fig. 18.—Timbering in a longwall mine in a medium pitching seam of medium thickness.

Coal in this mine is hard, without any butts or faces, and with the system of mining described, produces 67 per cent. lump coal over 4 in., 9 per cent. 1½ by 4 in. egg, and 7 per cent. ½ by 1½ in. furnace nut, or a total of 83 per cent. domestic sizes.

Steeply Pitching Seams (over 22°)

A thick seam, known as the Henry Ellen or Mammoth, in the Cahaba field, is shown in Figs. 19 and 20, which illustrate the method used to

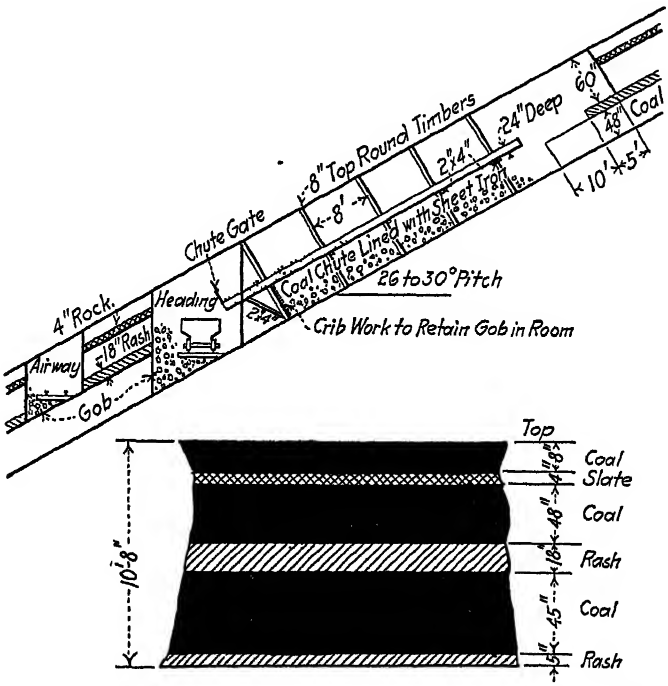


Fig. 19.—Cross-section through airway, heading, and room advancing, showing method of loading cars from room and disposition of impurities in seam.

mine steeply pitching seams. This seam is 11 ft. thick with 18 in. of rash parting about the middle of the seam, and 3 to 4 in. of slate parting near the top. The pitch of the seam varies according to location along outcrop and distance from the surface. The seam is steeper near the outcrop and flattens out somewhat as it nears the main fault, or limestone measures. The average pitch is about 26°. The seam is worked through a slope driven straight down the pitch with entries or headings driven at about right angles to the slope, from which rooms are turned straight up the pitch, as shown. In headings and airways, the coal is loaded directly into cars. Chutes are used in the rooms and cars are loaded from the chutes by opening the end gate. The cars are hauled

to the sidetrack at the slope by mules and hoisted to the surface by electric or steam hoists.

The coal is shot from the solid with permissible explosives, cutting shots being placed in the center of the working face in the top bench of coal and slab, or dependent shots, are placed at intervals of 3 to 4 ft. each way toward the ribs. After the top coal has been shot down and loaded, the middle parting of rock is removed with pick and shovel and then the bottom bench is drilled and shot up with a few light shots near the bottom of the seam. The top bench is carried 12 to 15 ft. in advance of the bottom bench in both headings and rooms. This seam gives off a large quantity of methane at the face of the coal, hence an efficient system of ventilation is required. Ventilation is effected with a motor-driven exhaust fan connected to the air shaft by an air duct with

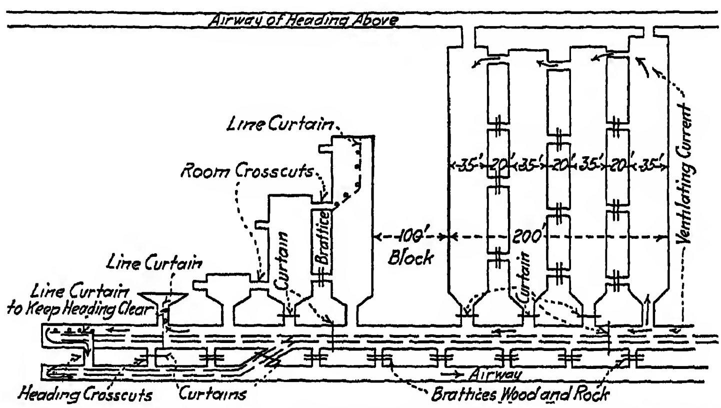


FIG. 20.—PLAN OF ENTRY AND ROOMS, SHOWING SYSTEM OF WORKING ADVANCING, ALSO METHOD OF VENTILATION OF MINES OF THE THIRD CLASS.

explosion doors over the shaft. The air intake is through the slope and manway and is conducted throughout the mine on a split system, by means of overcasts, brattices and regulators, constructed of rock, concrete or wood, according to the permanency desired. At times, it is necessary to use a line brattice or curtain from the last open crosscut (see Fig. 20), to the face of the working place and to aid ventilation. Electric cap lamps are used on this seam and the coal is shot by "shot firers" after the men leave the mine. An average miner will shoot and load from 12 to 20 tons of coal per shift.

In medium thickness seams of the third class, the difference in mining method lies in room depth and in handling rock on entries. There are no thin seams of this class mined in the state.

A semi-longwall method of mining has been applied in a few cases to thick seams steeply pitching; Fig. 21 illustrates this method. A slope was driven on the seam pitching 35°, which flattened in the lower workings to about 25°. On the first two entries, rooms were turned up

the pitch, the coal shot from the solid and loaded through chutes into 3-ton cars. This method was not a success, for the coal was almost completely shattered by the solid shooting, so a semi-longwall system was adopted. A section of this seam is in Fig. 22. Entries were turned so as to provide walls 200 ft. long. Alternate panels are left solid with the idea of advancing with one panel and retreating with the other.

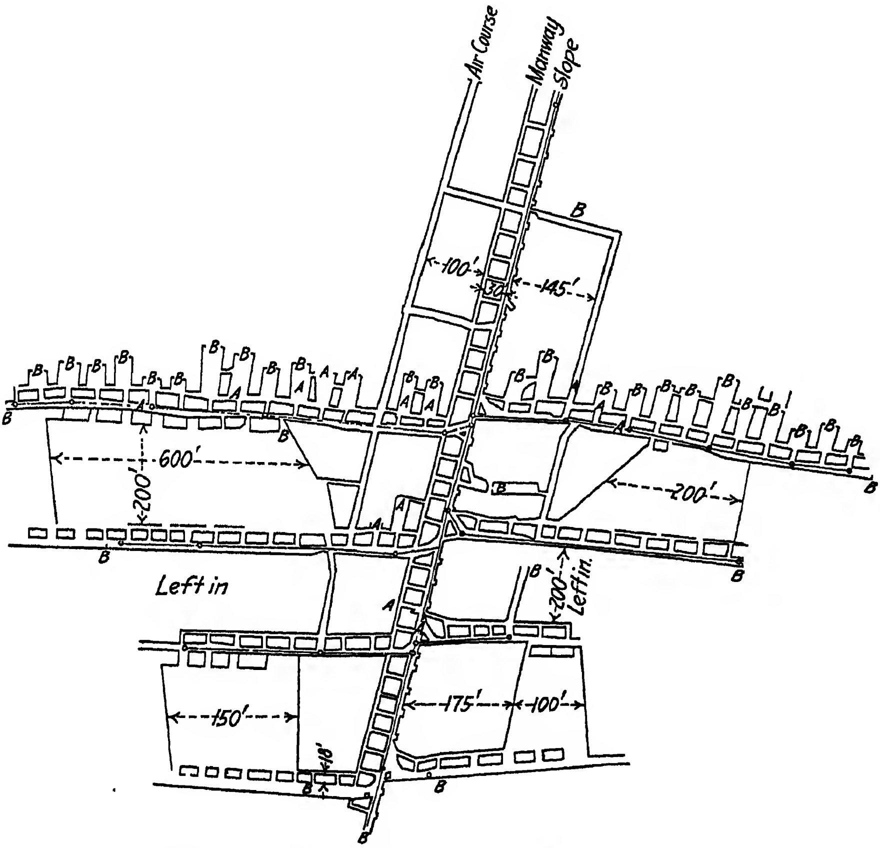


FIG. 21.—SEMI-LONGWALL MINING IN A THICK STEEPLY PITCHING SEAM.

The walls are undercut, beneath the bottom rock, with chain machines of the longwall type with a 5½-ft. cutter bar. These machines cut up the pitch. Each machine cuts about 100 ft., which is about as much as the men can "rock down" in a shift, in about 2½ or 3 hours.

In addition to the regular ropes on the machine, there is a safety rope by which the machine is lowered on the pitch after cutting. The rope is attached to a carefully placed face jack and winds around a drum on the rear of the machine; a friction mechanism attached to this drum is operated by the machine runner. When cutting on a pitch over 25°,

this rope is kept taut, as a precaution in case a feed rope should break; on pitches of 25° or under, the undercuttings in the kerf will hold the machine when a rope breaks.

As the wall advances, ordinary mine timbers about 6 in. in diameter are set about 4 or 5 ft. apart; these timbers are left in place until they begin to show weight, when four rows of large breaking timbers, 10 in. in diameter and larger, are set to within 3 or 3½ ft. of the face. Then all the smaller timbers are removed and a break follows. After the first break, it is necessary to timber for additional falls, as the face advances from 50 to 100 ft. On the lower entry of each wall, pillars 18 ft. wide and 25 ft. long with 10-ft. crosscuts are left to protect the entry. When the wall advances beyond a crosscut, the chutes are curved into the last one left open. The coal is loaded into the chute by the miner from which it passes by gravity into mine cars.

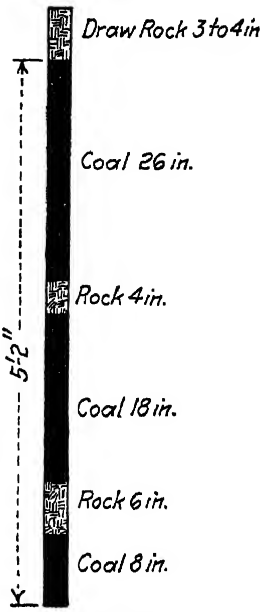


FIG. 22.

This system may, with conservatism, be said to be in an experimental stage. The operator anticipates some difficulty while retreating on the alternate panels. It is planned to drive a "raise" through these panels at a distance of 1000 ft. apart, and, if trouble develops while retreating, a wall will be advanced from a "raise" toward the face. This system is used on medium thickness seams of medium pitch. Where the pitch is under 18°, the chute is replaced with a conveyor, which consists, in most cases, of a chain traveling in a trough. The movement of the chain is sufficient to bring the coal to the mine car on the entry.

As an indication of the influence of the thickness of seams on average production per man, including company men, the following is submitted; the figures have been compiled by a large producing company and cover several months:

Thickness of Coal, Inches	Average Production per Man, Tons
28–30 (machine mined)	2.16
36 (machine mined)	3.90
44 (machine mined)	4.35
84 (solid shooting)	4.65
42 (steam shovel)	8.57

Ventilation

Furnaces are still in use generally at drift mines where there is no power, the system is costly and inefficient. At practically all mines where

power is required various types of fans are used. At some drift mines, small fans of about 5000 cu. ft. per min. capacity are used; one fan furnishing air for one or two drifts. At larger mines, where methane is generated, there are usually two drives to each fan; one of the drives is generally a motor and the auxiliary drive is a steam engine, or a gasoline or kerosene motor where there is no boiler plant.

At No. 1 air shaft of the Woodward Iron Co. there is a No. 16 Sirocco fan driven normally by a 150-hp. motor; the auxiliary unit consists of a Fairbanks-Morse 100-hp. kerosene engine. This engine is direct-connected to fanshaft by a sliding jaw clutch. The motor is driven by a belt, the belt pulley being loose on the shaft with jaws cast in hub to engage sliding clutch. In case of power failures, it is only

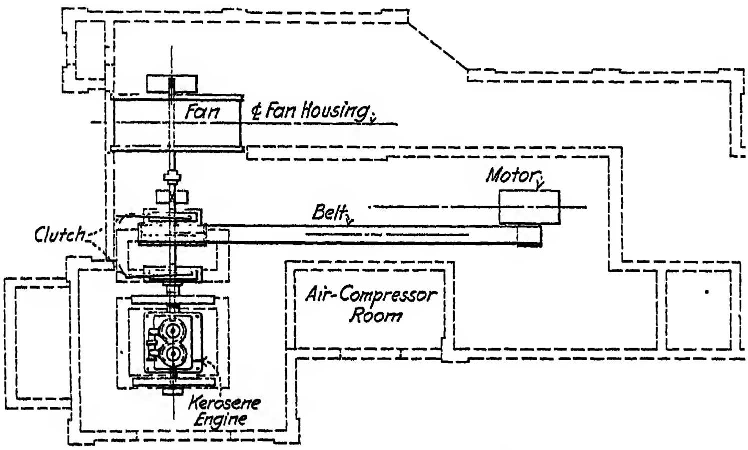


FIG. 23.—AUXILIARY DRIVE FOR MINE FAN.

necessary to throw out the clutch on the belt pulley, throw in the clutch to the engine and start the engine. The engine is started with compressed air from a receiver, which is always kept up to pressure by a small compressor that is driven by a 9-hp. gas engine. The entire change can be made in less than 5 min. Storage batteries are provided to furnish lights for the attendant of the fan in the case of power failure. The layout of this auxiliary drive is shown in Fig. 23.

Both continuous and split system are in use in the mines of the state; in the larger and better equipped mine, the split system is prevalent. Concrete overcast and stoppings made of mine rock, faced with a mixture of clay and cement, are found in the better class mines.

PUMPING

It has been found that as pillars are removed, particularly in the Warrior field, the cost of pumping increases greatly. In some of the

Pratt mines, where pillars were robbed as the entries were driven to the boundary, from 10 to 15 tons of water were pumped to each ton of coal produced. It is now the custom, at the larger operations, to extract not over 50 per cent. of the coal as the mine advances and large thick pillars are left; these are robbed as expeditiously as is practicable toward the end of the life of the mine. The reduction in pumping by this method will be great.

In the Cahaba field, where all the seams pitch, pumping is a great item mainly because there are conglomerate and porous strata above the coal seams. The problem is simplified, however, by the ability to drain the water to a common point.

Timbering

The methods of timbering vary greatly according to the top, and no set principle is followed. The cost of timbering varies from 1 cent per ton to 7 or 8 cents, depending on the nature of the top and the accessibility of the timber. In recent years, the cost of timber has increased, through the exhaustion of the timber supply. It is not uncommon at some mines to have the timber shipped or else hauled 10 to 15 miles in trucks.

Haulage and Tracks

In the larger, better equipped mines, slope and main haulage track is laid on sawed or creosoted ties with 40 to 60-lb. steel. On cross entries, the rail with mule haulage is 20 lb. and with locomotives 30 to 40 lb. The room rail varies from 12 to 20 pounds.

In medium and steeply pitching seams, as a general rule, mules are used on all cross entries for gathering coal to the slopes. Where the pitch is from 0° to 5°, locomotives are used in some instances. On flat seams, at all large mines, locomotives are used almost entirely for gathering and main-line deliveries. Storage-battery and gasoline locomotives are used at a few operations; and in a few cases endless and tail-rope haulage are found.

At one mine in Alabama, a belt is used for hoisting coal up a slope; this belt travels up a slope on a pitch of 19° 53′. The belt is 4 ft. wide, with troughing idlers on the top and straight idlers on the bottom. The belt centers are 485 ft. and the belt is kept taut by passing over an idler pulley located just under the head pulley to which a counterweight is attached. The coal is dumped in a one-car revolving dump underground. Under the dump is a hopper, which holds about 6 tons, or the capacity of three mine cars. Under the hopper is a shaker feed, which passes the coal from the hopper on to the belt. This shaker feed is perforated and the slack coal first passes to the belt and the lump coal is fed on the slack. The belt has a capacity of 2500 tons per day of 8 hr.

A 75-hp. motor drives this belt and the shaker screen. The advantages of this sytem are: reduced cost of all tipple and head-frame structures; reduced operation cost; reduced cost of hoisting machine—a 75-hp. motor as against an expensive hoist of 500 to 700-hp. capacity; by regulating the feed at bottom of the slope coal can be fed on to the shaker screen with such regularity as to make the screening more effectual; all danger from slope trips is eliminated.

This belt has been in operation about one year and no noticeable wear has been noted. The mine has not been brought to its full capacity, but 1100 tons have been handled over this belt in one day with the belt standing idle about one-half of the time.

Mine Cars and Dumps

With the various type and size of seams, many types of mine cars are in use in the state. In the more recent development, cars without end gates and revolving dumps are becoming the general practice. At one operation, on 25° to 35° pitch, Griffith bottom-dump cars are being used with success. One type tipple at a slope mine, illustrating the use of revolving dumps and mine cars without gates, is shown in Fig. 24. This tipple handles about 2000 tons per 8-hr. day with a counterbalanced haul of five cars per trip. The capacity of the cars is 3500 lb., making about 9 tons per trip.

The coal is pulled from the loading yard through a 30° slope about 1100 ft. long with an 800-hp. Vulcan hoist. Hoist drums are 10 ft. in diameter with a 4 ft. 6 in. face and a rope speed of 1800 ft. per min. A 500-ton reenforced-concrete bin, located over two standard-gage railroad tracks, receives the coal under the gravity dump. The bin is 39 ft. high on the approach end and 49½ ft. high on the hoist end; this gives an angle of about 8½° in the gravity dump, which facilitates the prompt return of cars by gravity as they are dumped.

The hoist operation is controlled by electric signals from the man on the top of the tipple, when cars are ready to be returned to mine, and from the man in the loading yard, when the trip is ready to be pulled out. The coal is weighed at the loading yard and pulled into the dump which is released by operator on the tipple. The dump is rotated by the weight of the coal. As the cars empty, the dump returns to its normal position and latches automatically, and is then ready for the next trip. As the coal leaves the cars in the dump, part of it is caught by sampling chutes in the top of the bin.

Every precaution has been taken to make the tipple as fireproof and safe as possible, the structure being reinforced concrete and steel; at the top of the tipple, on the hoist end, a 20,000-gal. water-storage tank is provided for fire emergency.

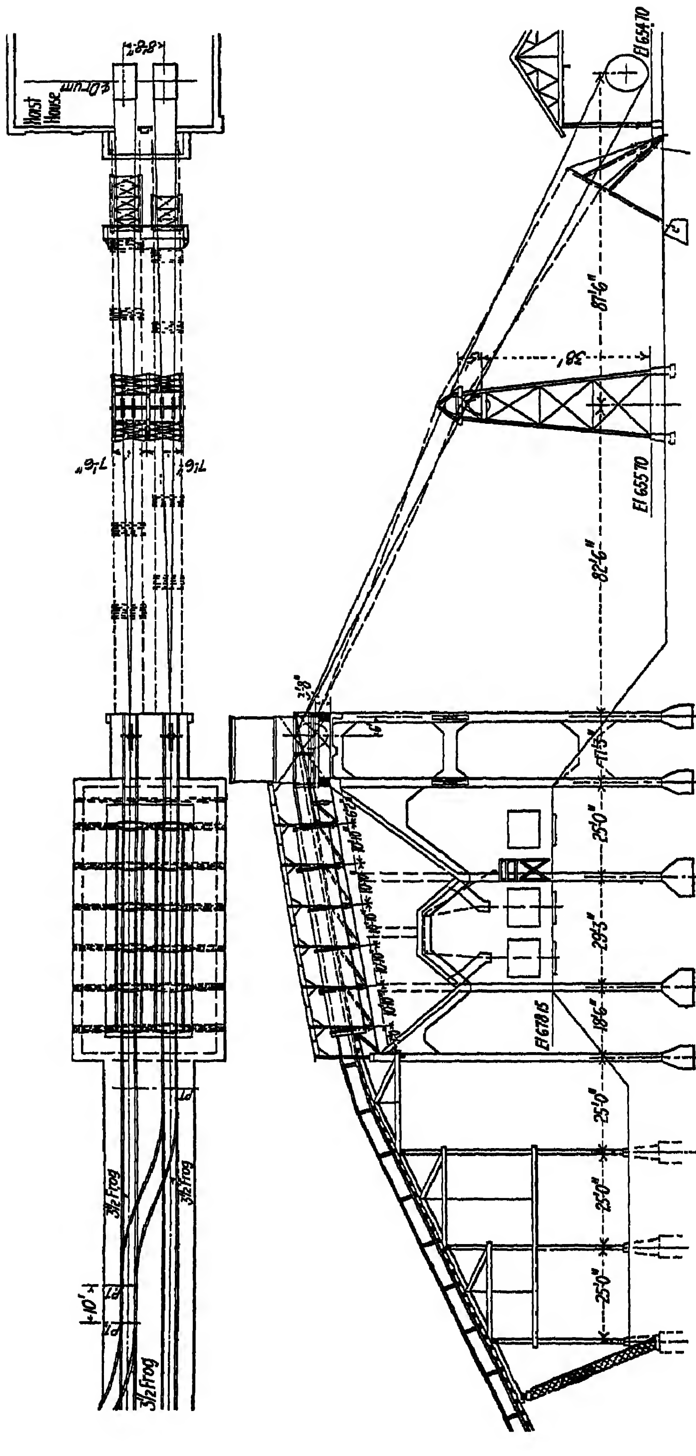


FIG. 24.—TIPPLE LAYOUT, DOLOMITE NO. 3 MINE, WOODWARD IRON CO.

Mine-run Sampler

All mines in the Alabama district have a dockage system, whereby miners are penalized for loading excessive quantities of slate or rock. The method generally used to check up the miner is to pick a few cars at random, unload them carefully, a shovelful at a time, and inspect closely the entire contents of the car. The weak point in this method is that only a few cars can be inspected during one day's operation and the miner, knowing this, will take the chance of not being discovered.

Erskine Ramsay, First Vice-President and Chief Engineer of the Pratt Consolidated Coal Co., has perfected a mechanical sampler that has made possible the testing and inspection of a large number of mine cars without actually unloading their contents.

The Ramsay sampler has been used more extensively by the Woodward Iron Co. than by any other concern in this district. This company installed its first sampler in May, 1920, at Dolomite coal mine No. 1. Since then, two additional machines have been installed at this mine, three have been installed at Dolomite No. 3, and four at the Mulga mine; this gives them a total of ten samplers.

At Dolomite No. 1, the coal is hoisted in four-car trips and is dumped four cars at a time by means of the two Ramsay rotary multiple slope dumps installed in parallel. Messrs. Crockard and Best, of the Woodward Company, made such changes in the usual style of these dumps, which are generally power driven, as to make them rotate by gravity. Installed under these rotary dumps are chutes having hoppers at the top; beneath the right-hand dump, there are two of these chutes by means of which the coal is directed to a sampler room, in which one sampler is installed. Beneath the left-hand rotary dump, there are four chutes leading to a sampler room on that side of the tipple, in which are installed two samplers, each sampler being served by two chutes. The chutes have a hopper arrangement at the top for receiving the coal samples and are equipped with rotary gates operated by cables, which pass over sheaves and extend to a position convenient for the sampler operator, who is able to draw the samples one at a time as required for convenient operation (see Fig. 25).

In the sampler, as installed by the Woodward Company, the operator releases the sample from the chute on to a double-deck shaking screen which is 24 by 42 in. and feeds slowly to a picking table 24 by 38 in. The upper of the two shaking screens has 1-in. diameter perforations and the bottom screen ½-in. diameter perforations. The lump material that passes over the upper screen is hand picked and then the material from the lower deck is discharged upon the picking table through the small gate and is picked. The fines passing through the lower screen go directly into the coal car and are not considered in the sampling, as

the removal of this material facilitates the operation and makes more careful and faster picking possible.

At Dolomite No. 3 mine, practically the same arrangement has been installed. Here the coal is dumped by two Ramsay rotary dumps in trips of five cars, instead of four cars as at the Dolomite No. 1. Chutes of the same design lead from beneath the coal dump to the sampler room in

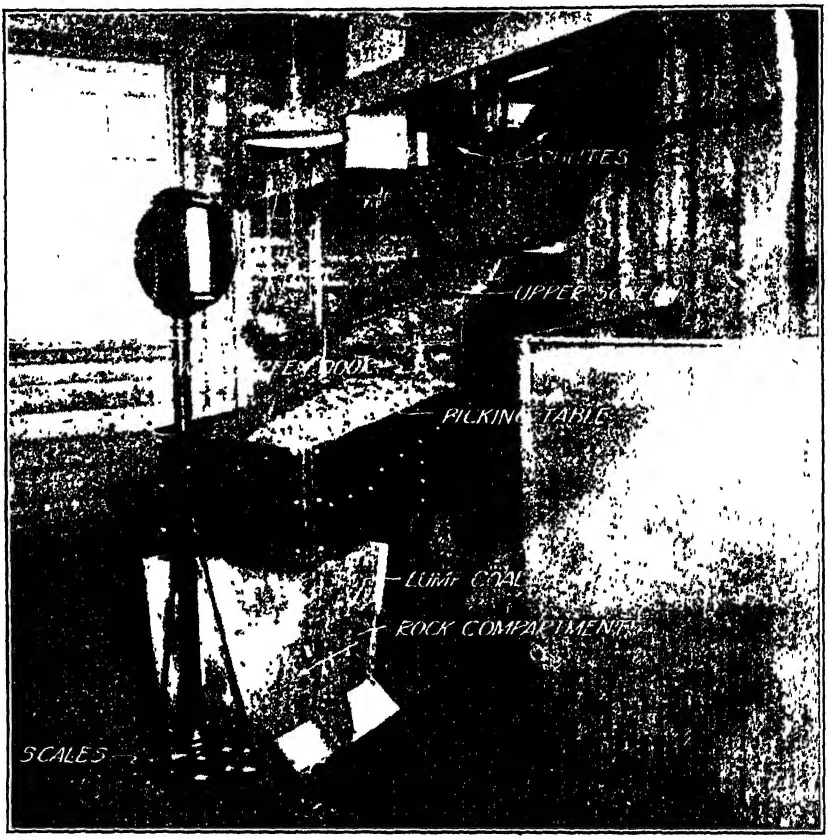


Fig. 25.—Ramsay mine-run sampler at Dolomite No. 1 mine.

exactly the same manner as at Dolomite No. 1 and are equipped with gates so that the coal may pass directly to the sampler or, if desired, may be retained to suit the convenience of the sampler operator.

At Mulga mine, four samplers have been installed. These are arranged in two lines back to back, but staggered to allow sufficient space for the attendants. The layout here presented unusual difficulties because of the small space available for the installation. The amount of head-room was so small that it was not possible to install gravity chutes; therefore, a small motor-driven carrier was designed to carry samples from

the receiving gates to any of the four machines. This carrier is a small bottom-dump car with wheels running between flanges on parallel 10-in. channels. A tripper is set by the operator at any of the four machines and when the switch is thrown, the car runs down, strikes the tripper, and automatically dumps, thence returning to the receiving gate. A single loading point is possible at this mine as cars are hoisted singly and dumped by a one-car rotary dump.

This battery of samplers is the latest installation made by the company and several small improvements have been made in the design. The shape of the skirt plate along the picking table has been altered and both shaker and picking table have been increased in length. The width of the machine remains 24 in. but the shaker screen is 48 in. in length and the picking table is 54 in. The method of actually picking the slate is the same at all installations; that is, the lump is first picked as it passes over the upper deck of the shaking screen on to the picking table. At the lower end of the picking table, there are two compartments; the coal compartment is adjacent to and immediately under the apron of the picking table and the slate compartment is just beyond the coal compartment. The operator lifts the slate by hand from the table and drops it in the far compartment, in the meantime the coal gradually passes off the picking table into the near, or coal, compartment. After the slate is picked out, the operator hastens the movement of the coal by brushing it into the coal compartment with his hands. As soon as the lump coal has been picked, the small gate retaining the small nut coal on the lower deck of the shaking screen is opened and this coal passes on to the picking table and is picked, the materials being placed in the proper compartments. When the sample is completely picked, the total weight on the scale dial is noted. A gate at the bottom of the coal hopper is then opened, allowing the coal to discharge into the car and the weight of rock is noted. The rock is then discharged, the gates are closed, and the weigh hopper is ready for a new sample. To facilitate weighing, dial scales are used with posts 6 ft. high, thus bringing the dial to the level of the operator's eye. These scales have a 13-in. diameter dial and are graduated to read half pounds.

This detailed operation applies to the Dolomite No. 3 mine, where the average sample weighs about 70 lb. and requires approximately 7 min. to pick.

The samples are identified by means of the miner's check number, which is fastened to the bottom of the car and is noted by the tipple man and passed down to the sampler man by means of a speaking tube. The sampler operator keeps a record headed at the top "Coal Sampler Record." At the left, on the next line, is recorded the name of the coal mine and on the right-hand side in this line is the date. Beneath this, in parallel columns, is the following information: First column, check

number; second column, total weight of sample; third column, weight of coal; fourth column, weight of impurities; fifth column, percentage of impurities.

The total cost of installing the sampler depends, of course, on the layout of the tipple and whether or not the coal can be dumped directly into a chute from the sampler or must be transported some distance. Also, the cost depends on the cost of supports for the sampler and sampler room, and other considerations of this character depending on local conditions. An estimate of the actual cost of one coal sampler complete built by the Woodward company in its Woodward shops, without considering the cost of chutes, sampler room, sampler supports, etc., was about $300. It requires from 3 to 5½ hp. to operate the sampler.

The scale is supported independently from the sampler, as otherwise the vibration would make close reading impossible. The Woodward company has found that the speed of the eccentric shaft operating the shaker screen should be about 150 r. p. m., also that the throw of the eccentric should be about 3 inches.

Figures furnished by the Woodward company show that the percentage of slate and other impurities in the coal prior to the time that the first Ramsay sampler was installed, as determined by washer loss, was approximately 20 per cent. of the total amount of material hoisted. Since the samplers have been in operation, the percentage of impurities, as determined by the washer losses, is about 10 per cent. The company believes there is no question but that the reduction in the amount of impurities by about one-half is due entirely to the sampler. It is not necessary to give actual figures as to dollars and cents saved. The amount of money saved is represented by the percentage of the total amount of material hoisted; the company has the 10 per cent. saving and a very important saving by eliminating the hauling, hoisting, and handling of impurities to the washers. Only one-half as much refuse now comes from the washers and is thrown away. Not only the cost of hoisting this material, but the wear and tear on the machinery and all of the other incidental things involved in the useless, unnecessary operation of loading rock and impurities are eliminated.

Safety

Considerable improvement is being made toward mining coal more safely in Alabama. A fair comparison of progress made in this direction is best visualized by comparing records for a 10-year period, say 1913 to 1923. In 1913, 73.7 per cent. of all coal mined in the state was done by pick and shovel methods and only 26.3 per cent. of the coal was produced by undercutting mining machines. For this same year, 68 per cent. of the coal produced was with permissible explosives and 32 per cent. with

black blasting powder. In the year 1923, 53.8 per cent. of all coal mined was done by pick and shovel methods, whereas 46.2 per cent. of the output was derived from undercutting mining machines. For this same year, 78 per cent. of the explosives used in coal mines were of the permissible type and only 22 per cent. black blasting powder. In brief, for the 10-year period, there has been approximately 20 per cent. decrease in the amount of pick and shovel work and a corresponding 20 per cent. increase in mining coal with machines. Permissible explosives show an increase of 10 per cent. for this period, while black powder showed a decrease of 10 per cent.

Marked improvement has been made in the matter of laying coal dust in the mines of the state. The principal methods now used is by the installation of pipe-line watering systems. In most of the large producing mines, efficient watering systems have been installed; while in some of the smaller operations, watering is done in a desultory way by use of water cars. However, a considerable number of small operations do not use any methods of laying coal dust to prevent its being raised into clouds with possible ignition and propagation.

In large producing mines, sprinklers are used to wash down the coal dust on ribs, roof, and along roadways, using 50 to 100 ft. hose of garden type. These methods have been supplemented, in a large number of the mines, by introducing water at the cutter of mining machines to wet the coal dust during the process of undercutting. It can be safely stated that fully 90 per cent. of the dust formed in our mines comes from the face, and it has only been in the last two years that attention has been concentrated toward laying coal dust at its source. Where coal is undercut with mining machines, without the use of water at the cutter head, large clouds are raised into the air; and while the machine cutting (bug dust) is being hauled, considerable leakage occurs through the cracks of cars and thereby increases the dust along roadways. Since the introduction of water at cutter chains, practically no dust is raised during the process of undercutting and the spillage and leakage of coal along roadways has been reduced to a minimum; in fact, to such an extent that there has been a considerable decrease in the number of road cleaners used before the introduction of water at the cutter bar.

The sprinkler systems in some of the mines in Alabama are among the best and most thorough in the United States. Pipe lines from tanks are laid from the surface or are connected with the discharge of pumps; water is then carried to the face of all working places, where sprinkling is performed by men especially provided for this purpose. Fig. 26 shows the method of sprinkling in one of the mines and illustrates the care with which the faces of the coal are washed down.

From experiments performed at the mines, under the supervision of the writer, it was found that twenty-one times as much dust is raised at

the face while coal is being undercut with machines as is raised by trips traveling at various speeds in the mines. It was ascertained further that five times as much dust is raised at the face while miners are loading coal as is raised by trips traveling in the mines.

At shaft bottoms, under former methods, great clouds of dust were raised during dumping operations; but since dust has been attacked at the face, practically no visible dust is raised into the air during dumping operations at shaft bottoms. The most striking change has been on surface tipples, where, under former conditions, tipples were enveloped

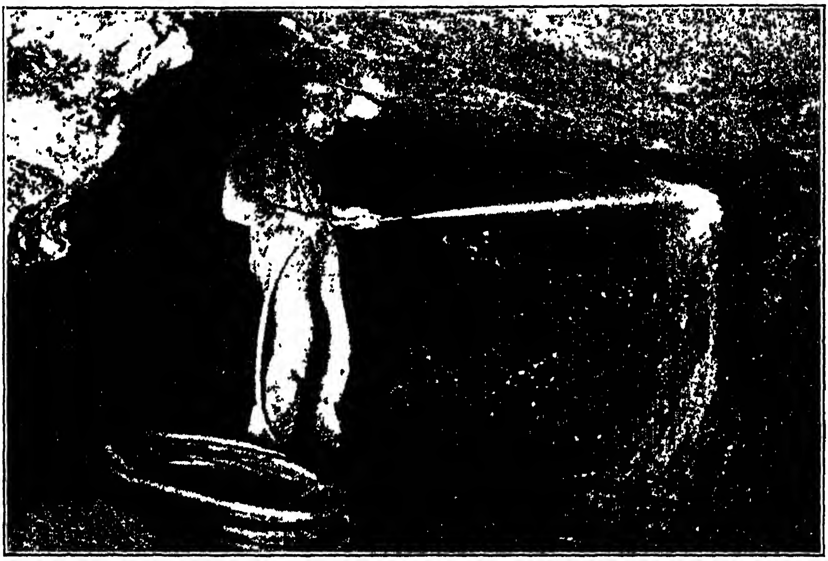


FIG. 26.—WASHING RIBS, FLOOR, AND ROOF.

during the greater part of shift with large clouds of coal dust but, at present, the atmosphere surrounding tipples is clear.

The cost of maintaining efficient sprinkling systems in Alabama is quite variable, depending on the output. It varies from a few cents to 6 cents a ton. The system, of course, is quite expensive to maintain, but it can be safely stated that, in the past few years, explosions that had originated in the interior of some mine have been stopped from propagating throughout the workings through the efficient system of sprinkling used.

Considerable interest is being manifested in this district at present in the use of rock dust as an explosion preventive. Insurance carriers, as well as the Federal government, have been active in the promotion of rock dust in this state. It is only the matter of a short time until even our most efficient methods of water prevention will be supplanted by the use of rock dust as an explosion preventive.

In addition to the strides that have been made through the introduction of permissible explosives supplanting black blasting powder, the substitution of mining machines for solid shooting methods, and the use of water for laying coal dust, together with the interest manifested in the use of rock dust, some of the mines have, within the past few years, introduced permissible electric cap lamps. The progress in this direction has been rather slow, but it is only a matter of a short time until a more extensive use of closed lights will be employed, especially in the gaseous mines.

Since the Bureau of Mines established a rescue station in the Birmingham district, several thousand miners have been trained in first-aid and rescue methods throughout the state, and it is quite rare to find a mine that does not have a large number of men trained in first-aid and safety methods and in the use and care of breathing apparatus. These courses of instruction have done a great deal toward educating the miner in safety practices. Some of the larger operations in the district have been carrying on the training of their employees in first-aid and rescue work from their own central rescue station. The Mining Department of the University of Alabama has provided a course in mining for mine foremen and the more ambitious men about the mines are availing themselves of this opportunity. This course covers from four to six weeks, and the charge, including board, is nominal.

In some of the mines in the state, mainly as a safety measure, section foremen are employed. These mines are divided into sections of several entries each. Each section has a foreman, who looks after the timbering, handling of explosives, and other safety measures, along with the production of coal; where this system is used, accidents are greatly reduced.

Some mines, in the employment of labor, use what is known as a contract system, under which the contractor is paid a margin above the digging price, and acts in the capacity of a section foreman.

In Alabama, as in other states, the more progressive and humane operators add to their cost per ton by utilizing precautionary measures, while the less progressive operators disregard ordinary safety standards. Accordingly, in periods of fierce competition, the progressive operators are at a disadvantage. While it is the judgment of the writer that the principle of "less government in business" is sound, at the same time, it must be conceded that government supervision is necessary when human life is at stake. The only means by which safety standards, applicable to all producers of coal, can be fixed, is through the action of the Federal government.

Labor

Alabama has made its own mine labor, which consists chiefly of native whites and negroes in the proportion of half and half. There

are some operations where 75 per cent. and more are negroes. The state has been non-union since 1908. There have been two invasions by the United Mine Workers of America—in 1917 and a final effort in 1920 and 1921. The union failed in its efforts to organize the state because of the loyalty of the miners. It has been found that best class of whites and negroes have no desire to unite in the same organization; this feeling is deep rooted and sincere.

The operators of mines in Alabama are not unmindful of the fact that under non-union operation there is added responsibility on their part that cannot be shirked. In the last 10 years, conscience has been awakened to the sense of duty that all men bear each to the other and more particularly of the employer toward the employee. In addition, the white man is beginning to appreciate the value of the negro based on the conception that after all each race has a common purpose and, in their own spheres, should work for the good of all. In the main, the old argument, so often advanced by union organizers, of "absentee ownership" does not apply to Alabama, where many mine owners live at their mines or else go to them at regular and frequent intervals. Close acquaintanceship, which means common sympathies and mutual aims, is, after all, the basis for the best and most lasting relationship of man to man.

Acknowledgment

In the preparation of this paper, the writer is indebted to Mr. J. J. Forbes, District Mining Engineer, Bureau of Mines, Birmingham; Mr. C. H. Nesbitt, Chief Mining Inspector, Birmingham; Mr. J. L. Davidson, Secretary of the Alabama Mining Institute, and to the operating officials of many of the coal mining companies of the state for their assistance.

DISCUSSION

Milton H. Fies, Birmingham, Ala.—There seems to be a great deal of objection (and I think it is justly founded) to government-owned business. The matter of state rights and paternalism enter into the question. But I do not see how a man who wants to do the right thing with reference to protecting his men in the mines can do it in competition with men who do not seek this protection. In one case, it is said that the safety insurance costs from 12 to 15 cents per ton. It is not at all difficult to imagine what would happen to a man who, during the past eight months spent 12 to 15 cents more per ton of coal than his competitors. The number of men lost in this region per thousand employed is higher than in any other state in the Union; our record per hundred thousand tons of coal produced is not as bad. That might lead one to conclude

that the introduction of mechanical contrivances and reduction of the number of men employed would decrease the number of men killed, but it will not always work out that way. We must get on the same basis if we protect our men.

R. V. Norris, Wilkes-Barre, Pa.—I do not agree with the author that safety is a national proposition. In Pennsylavania, and many other states, we have good safety laws and they are enforced vigorously. With the radically different conditions in the different mining regions and the different states, each state can probably take care of its own conditions better than they can be taken care of under a general national law. Further, our experience has been that the reduction in accident rate and consequent saving in the heavy compensation charges of the Pennsylvania laws more than make up for the cost of the safety campaigns carried on. It does not seem exactly right that a mine which does not need certain devices should be forced by a general law to put them in.

R. D. Hall, New York, N. Y.—The work being done to keep down the dust at the face is in the right direction. This work is going to make a considerable improvement in the conditions of the mines and is going to reduce the cost of rock dusting. Having one state compete with another and win out because its laws are not as strict and it is willing to run a few risks in the operations of its mines, affects us quite severely; it is necessary to have laws to keep everybody in line.

L. E. Bryant, Rockwood, Tenn.—About 15 years ago, a committee consisting of an Englishman, a Belgian and an Austrian came here to study American mining methods, and especially coal-dust explosions. One of these men said that the cost of safety measures they had been forced, by legislation, to install, doubled the cost of labor. That was the entering wedge of syndicating and nationalization of all their products; it had brought up inequalities in cost that would bankrupt certain syndicates unless they sold in a general pool. In America we cannot do that. As we get into deep-seam mining with its dangers, probably the legislatures will permit us to syndicate; then the high-cost mining can be operated at the expense of those who have not these high-cost operations.

George S. Rice, Washington, D. C.—In general, I believe the Alabama natural coal-mining conditions are more dangerous than prevail in other states. Some of the central states have a rather low accident record; this I do not think is because they take so much more precaution but because the conditions are not so dangerous.

As to the Federal jurisdiction over mine operations, I might say that as far as the Bureau of Mines is concerned that thought has never been entertained by its staff. The Bureau of Mines has no legal authority for police powers; it is a research bureau. Of course, that fact does not prevent some other federal agency being established by Congress if it is

constitutional, but I think that the majority of the people in the country object to federal interference with state rights Uniformity in the safety regulations is highly desirable and I am hopeful of important results from the governors' meeting that the President has called, to discuss the uniformity of coal mine regulations.[2] Certain things that are fundamental might well be covered by uniform regulations. The present regulatory system of Great Britain appears excellent. The laws are broad and there is uniformity required for the different districts on certain lines; for instance, ventilation, the combatting of coal dust, hoisting requirements, and features of that kind. In the matter of timbering, each district makes its own specifications but these specifications are agreed on jointly by committees of mine operators and the Mining Department of Great Britain. They have made a good beginning, as shown by the accident records since the enactment of the law of 1911. At that time the accident record was not nearly so good as in France and Belgium, which long held the lead.

On the public lands of the West, the Bureau of Mines does have some degree of control as the agent of the lessor, through the authority of the Secretary of the Interior. Under the leasing act of February 25, 1920, the Secretary, through the agency of the Bureau of Mines, was required to prepare leasing regulations that did not override State regulations. If they do conflict, the State regulation on the subject controls. The Department regulations cover not only safety features but conservation of coal in mining. It is too early to say what the effect is, as less than 2,000,000 tons per annum are being mined on the public domain. In those districts the accident rates in the past have been high; the steep pitch of some beds, and in some mines much firedamp, add to the ordinary dangers of mining.

In the matter of explosion prevention, Alabama is taking a lead in the use of sprays on the cutter bars of mining machines. Utah made this practice a requirement after the Castlegate disaster of last year. Much dry coal dust may be kept off roads by spraying the cars. That practice is also required in Utah.

The Bureau of Mines engineers have lost confidence in human ability to maintain effective watering throughout a mine at all times for the prevention of coal-dust explosions. This conclusion is not based on theory, but on statistics, observations, and tests.

To those of you who have not been following the progress abroad, I may say that Great Britain, France, and Germany have been going through the same stages we have. In Great Britain, the use of rock dust is now compulsory in all bituminous-coal mines. At first, the regulations excepted mines that were naturally wet. That brought up the question

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[2] This meeting was subsequently postponed indefinitely.

as to what was meant by "naturally wet;" and as explosions occurred in the "naturally wet" mines the requirement was made that the wetness of dust on floor, ribs, and roof should be 30 per cent.; otherwise rock dusting is required in all except anthracite mines. In the northern part of France, chalk or shale dust is being generally used; in central France its use is being considered. In Germany, for a long time they watered the mines; and they had many explosions. Recently, they adopted rock dusting.

If the amount of coal dust that is daily being made is kept at a minimum, and a good deal of it comes from the coal that is being ground under the wheels, less rock dust will be required and the expense of rock dusting will be small. An additional advantage of rock dusting is the ease of illumination of the passageways so that the condition of the roof may be more easily seen and haulage accidents lessened in number from better lighting. Some who have adopted rock dusting have said that the gain from better illumination alone justifies the expense of rock dusting.

H. S. Geismer, Birmingham, Ala.—Alabama is probably the pioneer in wetting the bug dust at the face. The Tennessee Coal, Iron & Railroad Co. and the Sloss-Sheffield Steel & Iron Co. started the work about the same time. After they perfected the system and sought a patent, they found that the Sullivan Machinery Co. had a patent covering spraying dust at the face, but had never used it. At present, nearly all the large mines use cutting machines, and wet the bug dust at the face.

L. E. Bryant.—When rock dust is used, is there any attempt in the winter to replace the moisture in the ventilating air? In the summer, there is no trouble with the humidity, but in the winter I think you should replace the water in the ventilating air going into the mines, so it will not take what little moisture is left in the mines.

Is there some relation between the number of explosions in the winter, when the outside air has not its load of moisture, and in the summer, when the outside air is loaded with moisture? If the air forced into the mine is dry, as soon as it is heated to 60° it will be ready to absorb moisture out of the mine. We used to have practically all the bad explosions between December and February.

L. E. Bryant, Roberta, Tenn. (written discussion).—Years ago the Birmingham district, so called, was that part of the Alabama coal field within the zone freight rates and switching charges; now the entire Alabama field can be included and, in the near future, Tennessee and Kentucky will contribute coal to this growing community. Charles Willard Hays,[3] speaking of the Pottsville formation, says:

Most if not all the rocks in the southern Appalachian field correspond in age with the lowest subdivision of the Pennsylvania series, namely the Pottsville, and south-

[3] U. S. Geol. Surv. *Ann. Report,* No. 22 (1902).

ward from the New and Kanawha Rivers it shows a decided thickening, which is very marked in the southern portion of the field and it appears probable that the 5000 or 6000 ft. of coal-bearing rocks in portions of the Birmingham District represent the same time interval as the few hundred feet in Western Pennsylvania.

After thirty years' residence in this field, during which time practically all the large modern mines between Ashland, Ky., and Tuscaloosa, Ala., have been developed, the writer believes that this statement is correct and that the southern coal measures are shaped like a wedge, thickest on the southern edge and thickening still more as it extends south under the coastal plain and fanning out to a shore-line deposit of one coal all along its northern, northwestern, western, and southwestern edge, where it raises along the highland geologically known as the Cincinnati Arch.

In the valleys of the four rivers that cut across this coal field from east to west, a cross section of the western edge of these measures is exposed and its coal veins have been opened and developed commercially; with the data now available, the progressive thickening southward is very noticeable.

Near the mouth of the Kanawha River, the Pottsville, or Lee, formation does not exceed 100 ft. in thickness and contains but one bed of coal, which is of little commercial value, but it does carry the well-known non-plastic fireclay.

In the Kentucky River drainage near Beattyville, the formation is about 200 ft. thick, with one and sometimes two beds of coal that have been worked at times; the best, known as the Beattyville, corresponds to the No. 1 and No. 1½ coals at the Stearns operations in the South Fork drainage in McCreary County, Ky.

At the mouth of the Rockcastle River, the Lee formation has thickened to 500 ft. and has three veins of coal of fairly reasonable persistence. The bottom vein rests on a fireclay just above the Pennington shales and is known as the No. 1 coal; it has a top split which is sometimes workable and known as the No. 1½ coal. From 45 to 90 ft. above this vein is a gas coal that is quite remarkable for its high volatile contents. Both of these coal horizons, the No. 1 and the No. 2, are below the big conglomerate of Safford which now becomes the rim rock of all the deep creeks and rivers. Here the whole ledge is 200 ft. thick and sometimes is all sandstone, but often the bottom part is replaced by slate, slaty shale or shaly sandstone. The top is almost always a level cliff sparsely sprinkled, conglomerate sandstone, forming rim rock cliffs wherever exposed.

Immediately on top of this, southward, after a few feet of shales and fireclay, occurs the No. 3 coal, another gas coal, in the Cumberland River and the South Fork region which is the coal worked on the C. N. O. & T. P. railroad north of Stearns, Ky.

At the crossing of the Kentucky-Tennessee line in the South Fork region, the Lee formation is 900 ft. thick and the No. 3½ and the No. 4 coal

horizons appear. The No. 4 coal can be considered the top coal of the Lee formation or the bottom coal of the Briceville shale series; its bottom split lies directly on the Corbin sandstone and extends across Kentucky from the Silerville mine, on the C. N. O. & T. P. railroad at the state line in McCreary County, to the Paintsville and Van Lear mines near the Kanawha River. The cannel coal horizons along this northwest outcrop are at the base of the series in the No. 1 and No. 1½ horizons, and again just above the No. 3½ vein, which is from 375 to 400 ft. above the No. 3 coal.

At Chattanooga, Tenn., the Lee formation appears to be 2500 ft. thick, and in crossing the state from north to south, appears to have gained 1600 ft. in thickness, whereas in crossing Kentucky it gained but 700 ft. This gain in thickness has been accompanied by a similar increase in coal; what at the Kentucky line is the No. 1 coal at Chattanooga has become a group of three irregular coals, all of which while thin have been worked in places and are known as the Castle Rock group and spread over about 200 ft. of measures, less on the western escarpment and more, if anything, on the eastern face of Waldens Ridge and Lookout Mountain.

The No. 2 horizon of the Kentucky line has become a second group of three very similar coals, known as the Dade Group. It also is worked at many places and all six veins are lower in volatile matter than the Kentucky or northern Tennessee coals of these horizons. All six are below the Safford conglomerate, on top of which, now 500 ft. thick, occur sometimes as many as four veins of coal, one of which, known as the Sewance, is nearly always workable. This group corresponds to the No. 3 coal horizon at the Kentucky-Tennessee line.

Still higher in the Chattanooga district occurs the Kelley-Durham group, which corresponds possibly to the No. 3½ and No. 4 coals in part of the Kentucky state line area. There are, however, low-volatile coals near Chattanooga. At Tuscaloosa, Ala., with its 4000 to 5000 ft. of coal measures, the Black Creek and Jefferson coals possibly correspond to the lowest, or No. 1, group of Kentucky and Tennessee; the Mary Lee or Horse Creek group to the No. 2 horizon of Kentucky and Dade Group of Chattanooga and the Pratt group of Alabama to the Sewanee—No. 3 group of Tennessee and Kentucky. The Cobb group possibly represents the No. 3½ horizon of Kentucky and northern Tennessee, and the Gwin and Brookwood groups, the No. 4 horizon in part in Kentucky and the Kelley-Durham group of Chattanooga.

The interval between the No. 3 and the No. 4 coals at the Kentucky-Tennessee line is 600 ft. with one thin coal; this may correspond with the more than 600 ft. of the Cobb-Gwin groups with their thin coals.

It must be kept in mind that any cross section of the Appalachian coal measures varies greatly from northwest to southeast and the Lee formation, as well as the other and richer measures on top of it, thicken from

west to east. It is, however, not known definitely what changes have taken place in the Lee formation in eastern Kentucky, where it disappears under the Briceville shale and is partly thrown up and exposed in Pine Mountain for the reason that no satisfactory drilling of this area or prospecting in Pine Mountain has been attempted. There is but one place known to the writer where one of these veins in the Lee has been traced southeast from the northwestern edge and the results are not entirely satisfactory because of an apparent split in the No. 4 coal, one of the key veins.

At Glenmary, Tenn., in Scott County, the No. 4 coal occurs at its best as to quality, but is very thin and has been worked for years. It rests on top of the Corbin sandstone which, at this place, is nearly all shale and shaly sandstone, except the top shelf rock. A drill hole proved it to be about 950 ft. above the No. 1 coal, which in turn rested on the Chester Shales so that the formation at that point was entirely regular.

Almost due south, about 25 miles, the Summers coal of the Flat Fork and Crooked Fork valleys occurs and there can be little doubt that this coal is the Brimstone Creek split of the No. 4 vein as it extends south and the bed rock on which it rests, and which has caused the Flat Fork and Crooked Fork valleys, is some ledge that has come in above the Corbin sandstone. The Harriman & Northeastern railroad climbs up this shelf out of the drainage of Little Emory River. In so doing it cuts across the outcrop of the No. 4 coal at the base of Little Brushy Mountain and also across the outcrop of the Brimstone Creek split of the No. 4, known here as the Stephens coal, at the top of its climb into the Crooked Fork valley; the difference between the two is 150 ft. vertically. The No. 4 coal here is the Poplar Creek-Coal Creek vein and is the coal worked at Briceville, just west of Coal Creek, and the coal on top of the Corbin sandstone at Briceville and at the base of the great shale to which the name Briceville has been given. No drilling has been done at Briceville, but there are evidences of coals thrown up in Waldens Ridge just east of town.

A well drilled for water at Petros, about 20 miles west, to supply the convict camp, passed through the No. 4 coal at about 400 ft. from the surface; the driller reported all of the lower coals in place, but the intervals and total distance given were materially greater than at the state line, showing a thickening of the measures to the south and east. The character of the drill used made any measurement of the coal veins doubtful.

The field is entirely cut through at Chattanooga by the Tennessee river, and the Tennessee and the United States Geological Surveys have covered this part of the field, as shown by the many openings of the mines formerly operating and still working, and shows the progressive thinning out of both coals and rocks from the eastern to the western shore line of the basin in Marion and Franklin Counties.

That part of the upper coal measures above the Corbin sandstone and No. 4 horizon does not extend south of the Coal Creek, Poplar Creek-Perros field, and while this field during the war supplied coal for the Birmingham district, and may one day furnish a large tonnage to Birmingham, as it is but 250 miles north and has a greater tonnage of cheaply mined coal untouched than the entire Alabama-Georgia field, this field can

FIG. 27.

hardly be considered as reserves for the Birmingham district today, although not only this coal but the whole Tennessee field and that part of the Kentucky field on the L. & N. and Southern railroads can be drawn on in any emergency, and special coals may one day be shipped regularly for special metallurgical processes.

In some of the deeper drill holes in McCreary County, Ky., and Scott County, Tenn., in which counties considerable oil well drilling has been done, about 50 to 100 ft. below the Chattanooga shale, and about 1000 ft.

below the No. 1 coal horizon, several isolated lenses of Clinton iron ore have been encountered, one of which at least is workable.

The small demand that existed for coal in the early days of Alabama's statehood was easily supplied from the northeastern end of the Coosa field and from that part of the Warrior field near Tuscaloosa. Today, the Coosa field, parts of the Cahaba field, and Lookout Mountain and Blount Mountain fields are better considered as a reserve. For the present the industrial development of the Birmingham district will have to depend on the Warrior coal field for its main supplies. Possibly, later, the lower cost of mining may let metallurgical coal from the L. & N. and the Southern railroad Kentucky and Tennessee mines compete with Alabama coals having lower freight rates but higher mining costs. When this stage is reached, possibly in 50 or 75 years, the probability is that most of the Warrior field and the plateau field and certain portions of the Cahaba field will be well along toward the peak of development before either the Coosa field or the lignite of southern Alabama and Mississippi will be called on. This latter fuel possibly exists in as great quantity as the true bituminous coal, but has never been systematically sought.

It has been estimated that the Warrior Coal field contains from 2625 to 4000 sq. mi. of coal, and most of it north of Tuscaloosa is not so disturbed or faulted that it cannot be worked. There has been, according to available statistics, mined and consumed to date less than five hundred million tons since the beginning of mining in the State, estimating the tonnage for 1924 at under twenty-five million tons.

The following estimates have been made by the state and national bureaus of the coal that may be recovered by the best methods in use, the fields being listed in the order of their importance and cost of mining. The small portions of these fields in Georgia are included.

	Tons
Warrior field	75,000,000,000
Cahaba field	7,000,000,000
Blount and Lookout Mountain fields	2,500,000,000
Coosa field	5,000,000,000
	89,500,000,000

In these estimates no thought has been given to the possible extension of the Warrior field to the south and southwest of Tuscaloosa under the coastal drift. This portion may be smaller than the absence of cross faults would indicate and on development it may prove to be beset with mining difficulties like the extension of the English fields into Kent under the chalk, and their probable extension at great depth under the Channel to a connection with the Belgian and the French fields.

With the great quantity of high-grade coal in sight, little thought has ever been given to southern lignite, which in all probability exists in the

southern portion of Alabama and Mississippi and some day will be utilized as fuel.

If the rate of consumption of Alabama coal should double every 25 years, which assumption has no solid basis, the estimated visible supply of 89,500,000,000 tons would reach exhaustion about the year 2110 or 2115.

	Tons
Mined to date	475,000,000,000
1925 to 1950	650,000,000,000
1950 to 1975	1,300,000,000,000
1975 to 2000	2,600,000,000,000
2000 to 2025	5,200,000,000,000
2025 to 2050	10,000,000,000,000
2050 to 2075	20,800,000,000,000
2075 to 2100	41,600,000,000,000
	83,025,000,000,000

We have, however, reason to hope that a way to utilize the full fuel value of coal will be discovered and that equal results may be looked for over the 750 years before exhaustion; and if we learn how to use the thinnest veins, Alabama bituminous coal may, with the hoped for continued progressive savings in operations, last 2000 or 3000 years, helped out locally with lignite, water power, etc.

New Orient, an Unusual Coal Mine

By George B. Harrington,* Chicago, Ill.

(New York Meeting, February, 1925)

This paper is a brief description of the design and equipment of a new coal mine in southern Illinois, which has many features not common practice in shaft coal mining and which is laid out and equipped for such unusually large capacity that it promises soon to become one of the largest mining operations of its kind. Shaft sinking for the mine, New Orient, was commenced in May, 1921, by the Chicago, Wilmington & Franklin Coal Co. In October, 1924, the plant was practically completed and was producing over 6000 tons of coal per day. There is good prospect that within a few years, when the mine is fully developed and manned, the output will attain a daily rate of 12,000 tons.

The coal seam at New Orient lies approximately horizontal, about 500 ft. below the surface, and may be said to run from 9 to 12 ft. in thickness. The tendency in Franklin County has been toward large mining units (5000 to 6000 tons per day) with great care being given to protection against fire and explosion hazards, and usually with quite elaborate surface plants permitting careful preparation of the coal into graded sizes comparable with those made in the anthracite region. The field is unionized, with uniform conditions of wages for various classes of work, hours, etc., and the Illinois mining laws are quite specific with regard to fireproofing and safety measures. These conditions make it difficult for any but well-equipped and well-operated mines to meet the competition of the numerous fine properties of which this county can justly be proud.

The coal company's property was already developed on the western side by the Orient mine—a much visited and written about operation (with which many of you are possibly familiar through the Department of Mines' film "The Story of Coal") which until recently held the 8-hr. production record with 8218 tons and usually ran quite consistently with production of slightly over 6000 tons per day. Experience at Orient was naturally one of the greatest influences in determining the location and character of the New Orient.

* President, Chicago, Wilmington, and Franklin Coal Co.

The first Orient was greatly handicapped in its early years by having been located several miles from any town and not on any through road or railroad; also by having been sunk on the edge of a hill just above the flood line of a wide creek bed. Much capital was required to establish a town and adequate labor supply and the plant site itself was cramped and expensive to adapt to the requirements of tracks and buildings. Like nearly all shaft mines in Illinois, Orient had one shaft through which the coal was hoisted, in the pit cars on cages, and through which men and materials passed; the second shaft was simply an air and escape way. The output in recent years had been limited chiefly by the capacity of the hoisting and screening plant, in other words, by the neck of the bottle, it having proved quite simple to keep as much coal going to the bottom as the shaft and screens could efficiently handle. Furthermore, the output was often curtailed by having to use the main shaft for other traffic than hoisting coal.

The area assigned to New Orient, adjoining Orient on the east, is roughly 2½ miles east and west by 4 miles north and south. The southern boundary extends about ¼ mile under West Frankfort, a mining town of 18,000, and, except for a little shoulder of high land just north of the town line, the surface of the property is below flood level for nearly 3 miles to the north. It was not practicable, therefore, to locate the shafts near the center of the area to be mined, as has been the usual Illinois custom, and a choice had to be made of the strip of high land near the southern boundary, or a location much farther north. Fortunately the southern strip was just about big enough for the mine plant and yards and was chosen because of its accessibility to man supply and transportation.

It was decided to use both main and air shafts for hoisting; the first to be equipped for raising coal only, and the second, or auxiliary shaft, in addition to the air compartment, to have balanced cages for handling men and materials, and also to be capable of hoisting coal in moderate capacity, for use during development years and on special occasions. The main hoisting shaft, which also carries the escape stairway, is approximately 9 by 24 ft. and the auxiliary shaft 14 by 32 ft., 130 sq. ft. of which is the air compartment.

By handling men and materials at the auxiliary shaft, it is possible to keep all traffic, except the day's run of coal, away from the main bottom; also the large hoisting unit of the main shaft is used only during the 8-hr. hoisting period of working days. The auxiliary shaft has comparatively small hoisting equipment so that, in effect, so far as the hoisting installation of the mine is concerned, there is a big mine with a small auxiliary for men and materials during actual working time, but only a comparatively small mine during the afternoon and night shifts, Sundays, holidays, and other idle days. With a maximum working-time limit of

six 8-hr. shifts per week, or only 48 out of 168 hr., this arrangement permits substantial economies over the common plan of using the coal-hoisting equipment for all purposes and occasions.

The mine is laid out on a panel system, which has worked well under the conditions of this field; see Fig. 1. The territory will be developed by two main entries, one running directly north from the hoisting shaft, about ¾ mile west of the east boundary line of the property, and the other paralleling the first about ¾ mile farther west. From these main roads, cross entries will be driven, 1650 ft. apart, and from the cross entries the

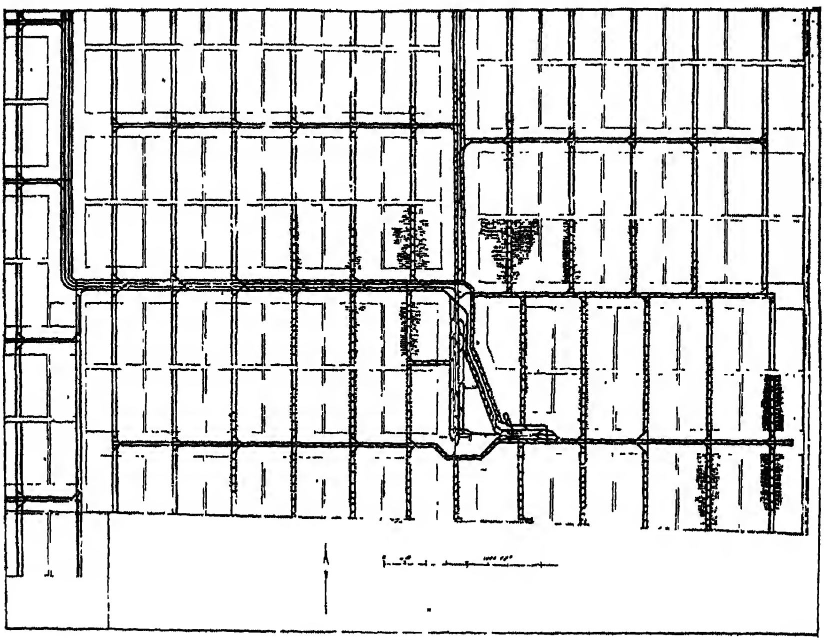

FIG. 1.

stub entries into the panels. The two main entries are being driven in sets of four, two of the openings being used for inbound and outbound haulage and the other two for air. A 70-lb. rail is used on main haulage, 60-lb. in the cross entries, and 30-lb. in panel entries and in rooms. Pit cars hold 6½ tons and are all steel, 42-in. gage, solid-end construction, equipped with semiautomatic couplers.

The main bottom is of usual simple type in plan, with get-a-ways and run-arounds on each side; but instead of running through the shaft, so that cars can be caged, the main entry runs to one side of the shaft and the cars pass through a rotary dump. The loaded cars go into the dump without uncoupling, couplers being of swivel construction, and two cars are turned over at each dump, their contents sliding into separate weigh

baskets located between the dump and the shaft. By dumping one car only at the start of a run, and two at each dump thereafter, and by emptying the weigh baskets into the skips after each has received and separately weighed two carloads of coal, there is an even feed of two pit car loads of

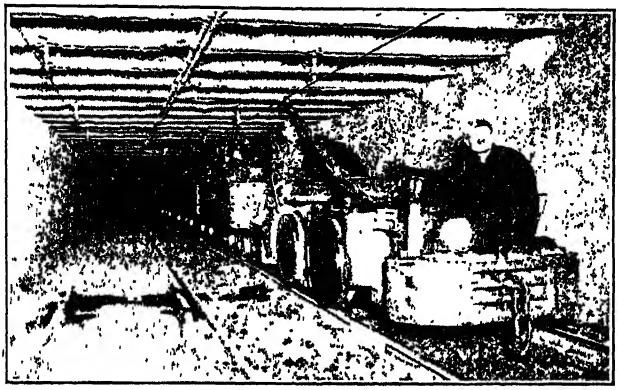

FIG. 2.

coal to each skip as it comes to the bottom while the other skip is discharging to the tipple.

Both shafts are concrete lined and the main bottom is supported by concrete walls and I-beams for short distances on either side of the dump.

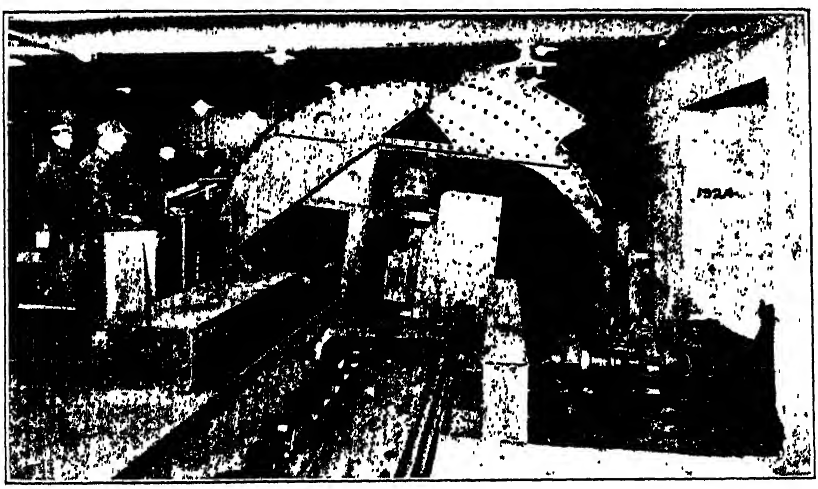

FIG. 3.

The haulageways and air courses are driven so as to leave a coal roof, which stands without timbering except in occasional broken places. All haulageways and air courses are rock dusted as a precaution against explo-

sions. Closed lights only are used in the mine. Great care was taken in the design of the rotary dump, weigh baskets and skips to avoid rough handling of the coal; it was aimed to have each movement of the coal a sliding one unchecked by any abrupt stops. Fig. 2 shows the main

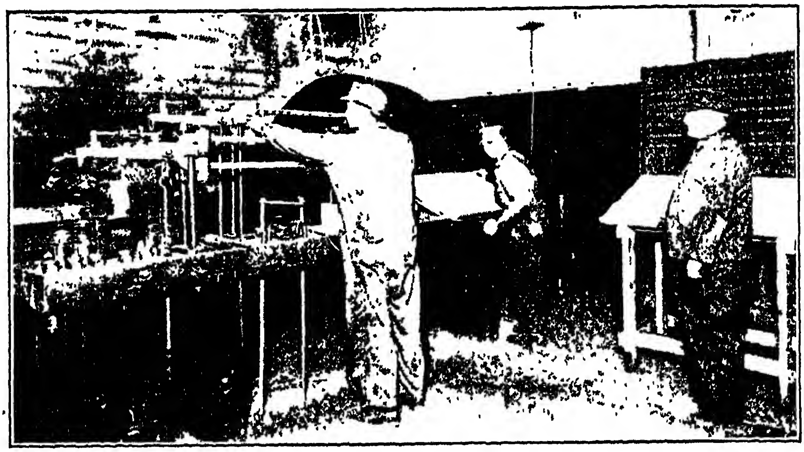

FIG. 4.

bottom, load side; Fig. 3, the rotary dump; and Fig. 4, the weighroom at the bottom of the main shaft. Fig. 5 shows an underground motor-generator room; Figs. 6 and 7 show a Myers-Whaley and a Joy loading machine at work in rooms, and Fig. 8 one of the skips entering the tipple.

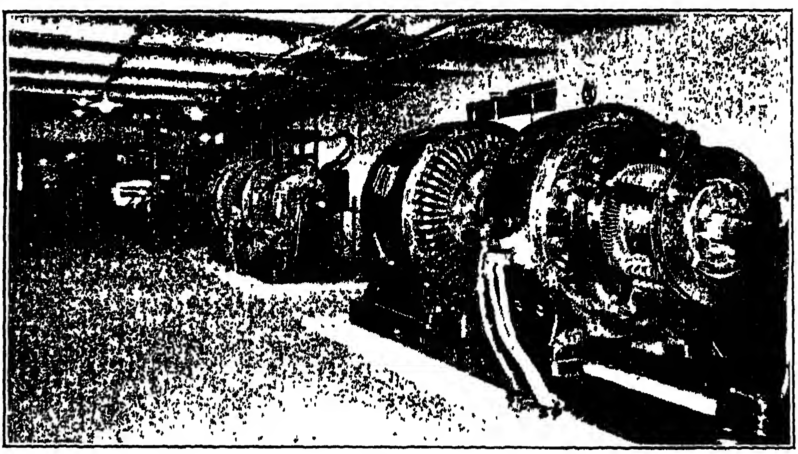

FIG. 5.

The mine is completely electrified, current for its operation being purchased from a large public utility company serving the district. Current is delivered to the mine substation at 33,000 volts, three-phase,

sixty cycles. Substation equipment consists of three 1667 kva., and three 500 kva., oil-cooled, single-phase transformers representing a total capacity of 6500 kva. Secondary voltage at the substation is 2300 and

Fig. 6.

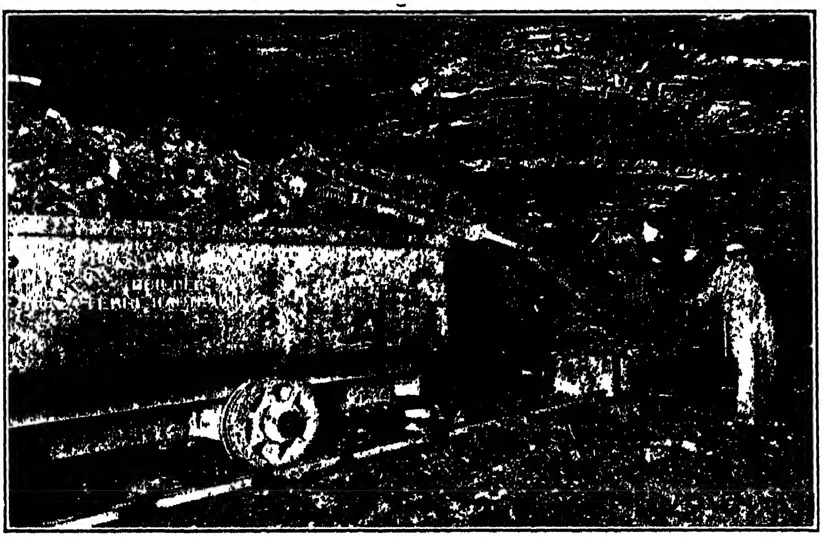

Fig. 7.

current at this voltage is conveyed through underground conduits to the switching room. From this switching room, which is in the auxiliary-hoist building, radiate the various circuits supplying current to the

several substations, both on the surface and underground. With the exception of the 33,000-volt incoming circuit, there is no exposed wiring in or about the surface plant.

All motors above ground, except hoist and fan motors, are 220-volt, alternating-current three-phase, sixty-cycle, induction type, and unit drive has been adopted throughout; stationary motors underground

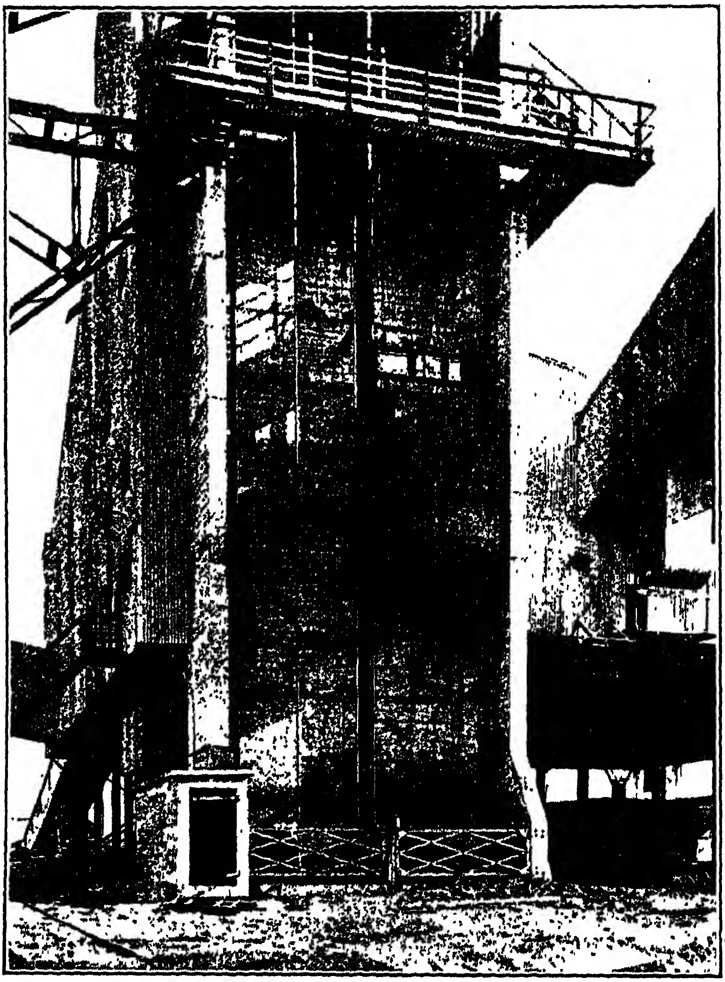

Fig. 8.

are 220-volt, alternating-current, three-phase, sixty-cycle, induction type. Motors underground driving coal-cutting machines, coal-loading machines, drills, pumps, fans, snubbers, compressors, etc., are 250-volt direct-current, supplied from an underground substation that contains, at present, three 50-kva., single-phase, sixty-cycle, 2300-220-volt, oil-cooled

transformers and three 300-kw., 1200 r.p.m. synchronous motor-generator sets 2300-275 volts.

Maximum 5-min. power demand, running at about 6000 tons per day, has been 1400 kw., with a monthly consumption, based on 24 working days, of about 430,000 kw.-hr. It is estimated that the maximum demand, when running at 12,000 tons per day, will be 2500 kw. and the monthly consumption, at full running time, in the neighborhood of 800,000 kw.-hr.

One of the most interesting problems of this mine was to provide hoisting equipment capable of handling the large contemplated output. With the normal 8-hr. production capacity calculated to be 12,000 tons and a vertical hoisting distance of 607 ft., it was necessary for the engineers to work out machinery for heavier duty than had before been done by this kind of apparatus.

The hoist installed employs the Ilgner system of equalization with modification of the Ward-Leonard system of control; the details are based on specifications and design of the company's engineers. The electrical portions were built by Westinghouse Electric & Mfg. Co., in coöperation with Nordberg Manufacturing Co., who made the mechanical portions. It has a cylindro-conical, step-up type drum 10 by 17 ft., located between and driven by two 2000-hp., 75-r.p.m., direct-current motors, ventilated type. Located in the same building is a flywheel set that supplies the d.c. current for operation of the hoist motors. This set consists of a 2200-hp., 2200-volt, induction motor mounted on a common shaft with and driving two 1650-kw., 600-volt, direct-current generators, one 50-kw., 250-volt, direct-current exciter and one 90,000-lb. flywheel running at 575 r.p.m. The slip is controlled by a liquid rheostat with the resistance varied by a torque motor and the equipment is designed to maintain practically constant input for hoisting on the normal cycle, the specifications for which were:

	Sec.
Average rest period	9
Acceleration	5
Full speed running	6
Retardation before entering dump horns	3
Retardation through dump horns	3
Total cycle	26

For cooling the hoist motors 30,000 cu. ft. per min. of washed air is delivered by a Sirocco No. 60 fan, driven by a 25-hp. induction motor running 575 r.p.m.

This big hoist had been in operation only a few months when this paper was written, and had not been driven at full rated capacity for more than short periods. But it has run very smoothly and satisfactorily, and has handled big tonnages when a steady run of coal was available

for short periods so easily, as to demonstrate that it will be able to meet all expectations when the mine is fully manned.

The auxiliary hoist house contains a geared hoist driven by a 200-hp. induction motor, and is provided (for special or emergency occasions) with a 400-hp. induction motor and a small pair of steam engines. A 200-kw. synchronous motor-generator set 2300-275 volt supplies direct current for night and idle day operation. This room also contains the mine ventilating fan, Jeffrey 5 by 12 ft. driven by a 300-hp., 2200-volt induction motor, and is provided with an 18 by 18 in. steam-engine drive for emergency service.

The auxiliary shaft cages are of the overturning self-dumping type, to permit the hoisting of rock, and occasionally of coal, in the solid-end mine cars. The auxiliary tipple has a rock chute and is connected by belt conveyor with the main tipple so that any coal hoisted here may be taken to the head of the screens in the main tipple. This conveyor is also utilized for taking to the top of the tipple any coal that may need screening after being recovered from ground storage.

The main tipple is equipped with two shaker screen units, each of well over 6000 tons per day capacity, feeding to picking tables and loading booms for the oversizes or to the screenings chute, from which the undersizes may be loaded in cars or conveyed to a rescreening plant for further subdivision. Both tipple and rescreener are of thoroughly fire-proof construction, electrically operated, and the arrangement of screens is such that great flexibility of sizing can be had.

The mine buildings are of simple brick and concrete construction. The main and auxiliary hoist houses, the shop building, the storeroom, and the wash house are of identical design, with Federal tile roofs. The mine offices are in a 40 by 50 ft. two-story building of similar kind.

Arrangement of the mine yards is such that the tracks for empty and loaded railroad coal cars and railroad switching are entirely separated from those parts of the yard where the men are passing to and from work and where narrow-gage tracks connect the auxiliary shaft with the shops and material yards. New Orient is served by four railroad systems, each of which reaches the mine yard over its own rails.

Quite a little experimental work has been done at New Orient toward the development of mechanical loading of pit cars underground. In fact, until September, 1924, practically all mine development work and production was accomplished with mechanical loading machines, and the general design of the mine and its large capacity were greatly influenced by the possibility that it may not be many years before mechanical loading in coal mines will be common practice. It is not intended that this paper shall go into the machine loading problem, but the experience here would indicate that the development of mechanical loaders will eventually permit important advances in coal mining methods. At the

present time, eleven Myers-Whaley shoveling machines and eleven Joy loading machines are used at New Orient.

As stated at the beginning, the purpose of this paper was to give only a general idea of the character of the New Orient undertaking, so that any one interested in any of its features may watch the progress of the operation through inquiry or a visit.

DISCUSSION

W. C. Adams, Chicago, Ill. (written discussion).—In designing the hoisting equipment for the Chicago, Wilmington & Franklin Coal Co., Orient No. 2, mine, the Ilgner-Ward Leonard system only was considered. The hoist has a potential capacity of 14,400 tons in 8 hr. with an expected average of 11,500 tons, which will make the operation the largest shaft coal mine in the world. The coal is hoisted in skips and the data on which the hoist was based are as follows:

Total distance of hoist	607 ft.
Maximum weight of coal per trip	26,000 lb.
Average weight of coal per trip	22,000 lb.
Weight of skip and yoke	15,100 lb.
Rest period	9 sec.
Total cycle	26 sec.

To fulfill the cycle, a maximum rope speed of approximately 4000 ft. per min. with a rope tension of 67,000 lb. for about 40 per cent. of the travel is required. This duty requires a carefully made rope and provision for the fatiguing action that will result from the high operating speed with the pronounced vibration that will be set up. A factor of safety of 8.1, based on the dead load, was used; a 2-in. rope of the modified Seale construction, using special strength plow-steel wires, was adopted.

The drum is of the cylindro-conical design having a minimum diameter of 10 ft., a maximum diameter of 17 ft. with a pitch of the grooves on the slope of 12 in. To take care of the side wear of the grooves, which is greatest on the small cylindrical portion, the groove pitch for this section is increased to 2¼ in., as compared to 2⅛ in. on the large cylindrical portion. In designing the drum, careful consideration was given to the proper proportioning of the metal in the various sections and the ribbing, to give strength and long life. Truss rods bring all parts of the drum shell in compression, thus giving the greatest strength possible to withstand the severe service expected.

The motor equipment consists of two 2000-hp. direct-current, force-ventilated motors, direct-coupled to the drumshaft, one on each side of the drum. Two motors instead of one were used in order to reduce the armature inertia and permit the use of commercial units. The use of forced ventilation made it possible to reduce the motor capacity required approximately 20 per cent., thereby reducing the armature inertia as well

as the cost of installation. Fan capacity for 35,000 cu. ft. of air per minute is provided, together with necessary air-washing equipment. The use of the forced-ventilated equipment made a net capitalized saving of approximately 6 per cent.

The motor-generator set, converting the power as received to direct current for the hoist, consists of two 1650-kw. 500-volt generators at 575 r.p.m., driven by a 2200-hp. wound-rotor type induction motor. A 90,000-lb. flywheel 12 ft. in diameter provides the necessary energy for equalization of hoist peaks and a 50-kw. exciter furnishes excitations for the generating and hoist motors. All units are mounted on a single built-up bedframe forming a four-unit six-bearing motor-generator set with flywheel.

The units are connected in order: generator No. 1 to motor No. 1, generator No. 2 to motor No. 2, thereby limiting the potential on any part of the equipment to 600 volts.

Slip regulator control equipment automatically varies the speed of the set with the load and permits the flywheel to hold the input from the line below a predetermined value; and when hoisting on the given cycle will hold the power input to within 5 per cent. of the mean.

The Ward-Leonard system of control is used and the generator field is varied by contactors controlled by a master controller and current limit relays, which function during both acceleration and retardation.

To maintain the cycle necessary to secure the large output expected from this mine, high rates of acceleration and retardation are necessary. The calculated rates of acceleration of the down-going skip is 13⅓ ft. per second. Tests on rates of acceleration place the maximum, without a dangerous slackening of the rope, between 15 and 19 ft. per sec. per sec. With current limit control of acceleration only, it would be possible to attain a rate of drum acceleration with light skip loads or empty skip that would be dangerous. To prevent this occurring, the field contactors are controlled in addition to the current limits by a traveling-nut switch, which regulates the rate of building up the generator voltage, thereby giving approximately constant acceleration rates irrespective of the load.

While the control is equipped with current limit relays to limit the pump back load in retardation, there is a question whether these should be depended on for retardation in emergencies when the maximum rate possible is desirable. The retardation time can be reduced, as compared to retardation with current limit relays, by the use of a Terril type relay, which will hold the regenerative braking current value to a constant, which may be the maximum capacity of the generators, or at a safe retardation rate.

In either case, if the main breaker goes out, control of the hoist must be maintained entirely by the brake. To take care of this emergency,

there is provided short-circuiting resistance controlled by contactors that will be immediately inserted across the terminals of the hoist motors with the opening of the main breaker. This resistance will give maximum safe load for the motors, which load will be maintained as nearly constant as is possible by two additional sets of contactors, which close successively when the motor-armature voltage is reduced by the slowing down of motor or drum speed to predetermined value.

Provision for prevention of overspeeding at any point in the shaft, for slowing down when approaching dumping point, for overtravel, and starting in wrong directions is provided in duplicate by traveling-nut limit switches, cam turn-off device, and limit switches with necessary back-out switches. Men are not handled by this hoist, therefore protection for man hoisting is not necessary.

In order to coördinate properly the operation of the bottom-dump station with that of the hoist to secure maximum hoisting speed, signals are given and certain gates are operated by the hoist-control devices.

To understand this feature, a description of the method of handling coal at the dump station is necessary. The coal is dumped by means of an air-operated rotary dump, directly into weigh pans. The weigh pans are manually operated and discharge the coal through a chute to the skips. At the end of this chute, there is a safety gate which is open only when the skip is on the bottom. These gates are operated by a crankshaft that revolves 180° in opening or closing. A motor is connected through a train of gears to this crankshaft. On the second-reduction gear shaft, there is a magnetic clutch for each gate.

Three lights in front of the operator, on the dump floor, show, respectively, when the skip is in the shaft clear of the safety gate, when the safety gate is opening or closing, and when the skip is landed and the gate is in the proper position for loading. When the descending skip has just passed the lip of the loading or safety gate, a contact on the traveling-nut limit switch releases the gate brake and sets the clutch to rotate the gate to open. When the gate is open, a cam switch on the crankshaft resets the brake and releases the clutch. This operation requires about 4/5 sec. The dump operator opens the weigh pan with the signal that the safety gate is opening. The weigh pan, in opening, energizes a time relay, which after an interval of 8 sec. gives the signal to hoist direct to the hoisting engineer. Similar operations control the closing of the safety gate. The operations are repeated for both skips so that the hoist works on a set schedule and delays are reduced to practically nothing.

Andrews Allen, Chicago, Ill. (written discussion).—There were many new problems to solve in the design of this plant, which is intended to produce an unprecedented daily tonnage of domestic coal, cleaned and sized for market requirements. Early in the work it was decided to confine ourselves, where possible, to units of moderate size and handle the

large output by duplication of units rather than by fewer units of unprecedented size. The only exception to this plan is the electric hoist, which probably is a little larger than any hoist previously built.

To defer unproductive expenditures, it was decided to build the auxiliary tipple in advance of the main plant and to develop the mine from this unit. This is common practice at big mines and it enables the operator to arrange an orderly construction schedule without congestion; also it takes care of the mine requirements as they arise.

The auxiliary tipple was located at the airshaft and was arranged for producing mine-run coal only, with a belt conveyor to the main screens in the main tipple. It would have been better for the immediate utilization of development coal to have introduced a set of screens in this tipple for preparing market sizes, but it was decided to sacrifice this small temporary advantage for the ultimate advantage of the plant. Another "screens and loading tracks" would have been a nuisance and an expense throughout the life of the mine, so preparation of this coal was deferred until the completion of the main tipple.

The various uses for the auxiliary tipple were carefully planned for the ultimate work of the plant. Cages of the overturning type were installed so that solid-end cars could be used throughout the mine, and arrangements were made for economical handling of the rock and mine-run coal into railroad cars. The belt with feeder to convey coal from the weigh pan to the main tipple was further justified by its use as a docking conveyor. It is the practice at this mine to hoist a certain amount of coal every day through the auxiliary shaft so that cars, picked at random or from special localities, can be carefully examined and docked. In practice, the coal is spread from the feeder on to the belt, which is then stopped, the coal picked over, and the docks removed; then the belt is started again and the coal is delivered into the feed hopper of the main screens.

This belt is also used for taking storage coal into the main tipple for reprocessing. For this purpose, a track hopper has been provided at the auxiliary tipple with feeders taking coal into an inclined flight conveyor which carries it to the belt. Coal hoisted at night is loaded into railroad cars and reprocessed in this way. The whole arrangement is so elastic that all conditions, foreseen and unforeseen, have been handled satisfactorily. The overturning cage[1] is handling a 6-ton car regularly, smoothly and rapidly and without excessive maintenance cost.

The mine cars are of all-steel construction. An extensive investigation of steel mine car construction, especially of the defects and failures, showed that the steel car had few friends and many enemies. It was decided, nevertheless, to work up separate designs for cars of steel, com-

[1] Andrews Allen and John A. Garcia: Skip Hoisting for Coal Mines. *Trans.* (1921) **66**, 370.

posite, and wood construction. Sixteen of these experimental cars were built, their costs were compared, and then they were tested by service in the mine. As a result of this study, it was decided to adopt a steel car, 3½ ft. high and capable of holding 6½ tons with a 2 ft. surcharge.

The construction of this car differs quite radically from most others. It is built on a double-channel sill, or bumping column, running from end to end, with the axles passing through it and the couplers attached directly at the end. The cars are convex ended, thus giving maximum capacity and maximum clearance between cars on curves. The bottom plate is ⅜ in. thick, so as to provide sufficient metal for resisting rust and sufficient thickness for driving rivets that will not loosen. The sides of the car are reenforced with heavy bars, instead of angles, or formed edges, and the ends have bar and angle reenforcements.

The running gear was especially designed, to allow as much flexibility as possible. Four or five makes of trucks were used on the experimental cars. Most of these trucks had self-alining boxes and all were carried on spring bearings. It was expected in this way to relieve the car, bearings, and rail from the hammer blow of a rigid car on a rigid track and also to neutralize the excessive stiffness of the car body and keep the wheels on the track by allowing each wheel a certain amount of independent vertical motion.

A test of each truck was made under running conditions and they were taken apart, studied and compared for strength excessibility, lubrication, and wearing qualities. As a result it was decided to adopt Timken bearings, set in the wheels with the shaft tight in solid boxes, flexibly mounted on the car, with rubber cushions instead of springs.

In the same way the question of couplers was settled. An invitation was extended to coupler manufacturers to submit full-sized samples for test and many companies responded. It was decided to call for a spring-mounted combined coupler and bumper of automatic or semi-automatic type. After five or six couplers were tested it was discovered that it was easy to make a coupler automatic, that the difficulty lay in convenient uncoupling and in bumping without coupling. One coupler would couple almost on any angle, but it would not uncouple easily nor bump without coupling. The study of these couplings resulted in the complete elimination of all but two, one an adaptation of the Tomlinson coupling, in general use on electric cars, and the other a semiautomatic coupling designed by ourselves. It was necessary, however, to redesign both of these couplings to meet the conditions actually developed in service. The final selection was made on the basis of actual test and price. Both designs stood up and did the work but the semiautomatic design cost only about two-thirds as much as the other and seemed to be automatic enough to fill the bill. This coupler is of a spring-bumper type. When the pin is cocked for coupling, the cars will couple automatically by

impact, on a straight track or one with a moderate curve. During the year and a half in which these couplers have been used, there have been no coupling accidents and almost no maintenance expense.

Taken in their various features, the cars are a wide departure from the general coal-mining practice. So far they have stood up remarkably well and have shown no weakness of any kind.

The main tipple was designed for making three prepared sizes, namely: 6 in. lump, 6 by 3 in. egg, and 3 by 2 in. nut. The screenings are either delivered to railroad cars or taken by belt to a re-screener, now under construction. The nominal capacity of the mine is 1500 tons per hour, with provision for a 25 per cent. excess peak load, making a total of 1875 tons per hour, of which the proportion of lump may be assumed at 350 tons; of egg at about 375 tons; and of nut at about 225 tons per hour.

It was evident that several picking tables would be necessary for each size of coal and very possibly several screenings. For instance, 375 tons of egg coal per hour would lie 6 in. thick on one 5-ft. picking table, running at 100 ft. per min. As effective picking could not be done under these conditions, it was decided to use two tables for each size of coal. These tables are 5 ft. wide with 30 ft. of picking space on each. The tables for egg and nut are rubber belts, running on flat Timken bearing idlers; the lump table is a single beaded flight conveyor. The speed of the tables was arranged so that it could be changed from 50 to 100 ft. per min. by using pinions of different sizes. The tables have run at 75 ft. per min., except the egg tables, which have been speeded up to 100 ft. per min. It may be interesting to note that better results are obtained at the higher speeds. The picking on the egg table is more effective at 100 ft. per min. than at 75 ft. The coal spreads thinner and the men naturally work faster at the higher speeds. At the same time exactly the same quantity of coal passes over the table whatever the speed, so the greater activity is a clear gain. We are aware that this is contrary to usual practice, but it is our conclusion.

The screening plant consists of a pair of 8-ft. pendulum-hung, center crank-driven shakers, each with its own apron feeder from the dump chute and arranged so that either screen can take the total tonnage from either skip. The screens are set parallel to the loading tracks and extend toward the auxiliary tipple. One screen is set 10 ft. ahead of the other so that each feeds every other picking table, thus giving successively a pair of lump tables, then a pair of egg tables, and then a pair of nut tables. Each pair of tables terminates on the horizontal run of a pan-conveyor loading boom, of which there are three, so that the product of two tables is loaded on one track. The curved chutes connecting the picking tables with the loading booms were carefully designed so as to carry the coal with minimum breakage.

Thus the tipple is built around a square, the coal is dumped and screened toward the east, picked toward the south, and loaded toward the

west. The control for the tipple (except screens and feeders) is located in the southwest corner where the operator can see all the various units of the tipple and can watch the loading from a most advantageous position. Besides handling the various controls, which are of the push-button type, he handles the car retarders, which control the movement of cars on the three tracks for prepared sizes. The arrangement is compact and accessible. The structure, especially the picking space, is well lighted. The floors and inside stairways are of concrete, the outside stairs of subway grating, and the covering of corrugated zinc.

The system of skip loading, in which an intermediate bin and measuring gates are dispensed with, is known as the "Haralgar" system, a name derived in part from the author of the paper, who had an important part in its development. This system has been frequently described and only the notable features at this particular plant that differ from other installations will be described. It has been the practice, when using this system, to allow the skip to operate the spill or loading gate by sword-arm levers and by chains or latches, all of which methods generally are satisfactory. In this case, however, the rate of acceleration and retardation of the skip was so great that it seemed doubtful whether any mechanical operator actuated by the skip could be made to stand up. It thus became necessary to devise a reliable interlocked operation without mechanical contact of heavy parts. The electric-operated gate devised has proved very reliable, rapid, and satisfactory. The design, also, made it possible to install a system of automatic signaling, which greatly aids smooth and rapid operation.

Briefly stated, the gates, which are of the undercut variety with lips projecting over the skip, are operated by a half revolution of a single-arm center crank, always in the same direction. The crankshaft is timed to open or close the gate in 1½ sec. for either operation and is operated by a train of gears from a motor which runs constantly. There is a flywheel on the motor-shaft so that the gate will operate for some time after power is off, and corresponds in this respect to the effect of the flywheel on the hoist. The pinion operating the main gear is engaged by a magnetic clutch, keyed into the first countershaft and a magnetic brake is set on the end of the crankshaft. When the skip has descended to a point below the lip of the gate, a contact is made with a switch, which immediately energizes the clutch and opens the gate. When the gate shaft has made nearly half a revolution, a cam contact on the shaft cuts off the power from the clutch and puts on the break. The reverse operation is performed by a cam on the weigh-pan gate, which makes an electric contact through a time-limit relay and starts the gate to close a certain number of seconds after the weigh-pan is operated. When the gate starts to close, a cam on the gate shaft makes a contact, which signals the hoisting engineer and the gate stops in closed position at 180° as before. There is also, of course, a

manual control for use in emergencies, but the regular operation is automatic. All current for operating the clutches and brakes is taken from the exciter set on the main hoist so that current is always available when the main hoist is running.

Colored lights indicate the position of each skip in the shaft—white when the skip is in the shaft, red when the gate is in operation, and green when the skip is on the bottom. The operator never sees the skip, as the shaft is entirely closed off at the dumping point; but he can operate the dump rapidly and safely by means of the lights.

It would be difficult to find a finer example of coöperation than in the design and construction of this plant. The author, his operating officials, chief engineer, and ourselves have from the start discussed all questions of layout, design, and construction in "a committee of the whole."

Graham Bright, Pittsburgh, Pa. (written discussion).—I have visited the Orient No. 2 mine and can say that it seems that nothing has been omitted to make this the largest and most efficient coal mine in the country. Everything has been done on a large scale, and at no place does there seem to be any cramping of space. At the time of my visit, the output was about 6000 tons per day, working only one skip. From the deliberation of the few men in the tipple and at the bottom one might get the impression that the mine is not in regular operation at all and that only a small amount of coal is being hoisted. The most striking feature of the entire installation is that, at all parts of the system, there seems to be some excess in capacity provided. The complete preparation of the coal is also an outstanding feature in these days when the public is getting very particular in the purchase of coal.

I am particularly interested in the hoisting equipment, as I had the privilege of being associated with the estimating, selecting, and building of the electrical equipment. The installation of this hoist is an epoch in electric hoisting as it is more than twice as large as the next largest electric hoist in any coal mine in the world. The southern Illinois coal field is a field of high-speed hoisting, and records already high are constantly being broken. The capacities and rope speeds represented by the Orient No. 2 hoist are so much above what is considered good and safe practise in other fields that only a pioneer would dare select them. The results obtained seem to amply justify the selections.

In the selection of the electrical equipment, no new types of equipment were involved. The hoist motors, generators, alternating-current motor, and a large part of the control had been built before for rolling-mill service, which is similar but more severe than hoisting service. The capacity of the equipment is such that the maximum weight of 13 tons per skip can be handled at full speed without overloading.

The installation of the hoisting equipment is worthy of note. The flywheel set is located in a room separate from the hoist, and the wiring,

busbar, and cable installation are of the very best. It would be hard to imagine how any trouble could occur from faulty installation. Large, slow-speed hoist motors must be furnished with forced ventilation; this is the first installation of a coal-mine hoist, to my knowledge, in which the hoist motor is furnished with washed air. Such care and precautions mean a long and uninterrupted service from the equipment.

The question naturally arises whether the equipment at Orient No. 2 is the last word or whether in a future installation with similar conditions to meet some change would be advisable. The experience in this country has been that when a large pioneer installation has been completed it is watched most carefully, and, within a short time, some slight changes become apparent which would make a future installation a little simpler, a little safer, a little more economical, or a little less expensive.

Recent improvements in drums, skips, and pit-car design bring up certain features in connection with the hoist that should be considered in a future hoist of similar conditions. The first is that of drum shape. Where very high rates of acceleration are involved, the cylindro-conical drum is sometimes at a considerable disadvantage on account of its relatively high inertia. This is particularly true with a geared motor, as the cylindro-conical drum is necessarily of a slower speed than the cylindrical drum. The cylindrical drum can be a single drum with very little idle space, and its inertia can be kept relatively lower than the cylindro-conical drum. There is more reason in selecting a cylindro-conical drum with a steam-engine drive than with a direct-current motor drive, as the steam engine has a definite maximum torque while a motor does not.

In the case of Orient No. 2 hoist a cylindrical drum would increase the size of the hoist motors and generators about 15 per cent., and the actual power required about 3 per cent. The only advantage gained would be a reduction of the maximum rope speed from a little over 4000 to 3360 ft. per min. The wear on the rope, however, would be very much less with the cylindrical drum, and as the drum and motor speed would be from 25 to 50 per cent. higher, the actual cost would be less than with the cylindro-conical drum. Ordinarily the cylindrical drum is selected as the same size as the minimum diameter of the cylindro-conical drum. In this case, however, the cylindrical drum was selected at 12 ft. in diameter as the maximum desirable for 2 in. cable.

The second change would be the consideration of a larger capacity of skip. In the study of skip hoisting, it will be found that for shafts up to 600 or 800 ft. deep, when the rope speed exceeds 2000 to 2500 ft. per min., the installation is penalized by relatively high first cost, high power consumption, and less safety of operation. The scheme at Orient No. 2 requires two car loads to each skip and each car holds an average of 5½ tons. At the time the scheme was laid out this was considered a very large car, especially for hand loading. Since that time car design has

advanced and mechanical loading has come to the front, as mentioned. At present, an 8- or 10-ton mine car for conditions similar to Orient No. 2 is considered feasible, and can be designed for either hand or machine loading. Of course, there are several methods by which a skip can be loaded to any desired capacity, but let us assume that 8-ton cars are available and that the skip will hold 16 tons. For a capacity of 1500 tons per hour with an 11-ton skip, a trip must be made every 26 sec., 9 sec. are required to load, 6 sec. to slow down (this could be reduced to 5 sec.

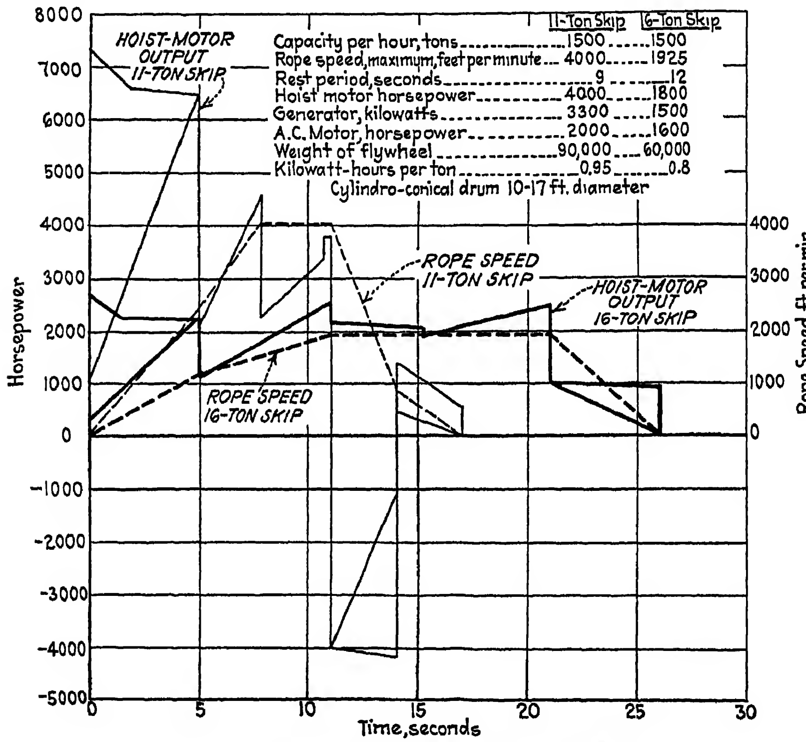

FIG. 9.—HOIST CYCLES, NEW ORIENT MINE.

except that it is necessary to slow down for the dumping horns). The actual operating time is 17 sec. and the maximum rope speed is a little over 4000 ft. per min. The maximum power required during acceleration is close to 8000 hp., and during retardation there is a maximum braking of 4000 hp., the total variation in the cycle being 12,000 hp.

With a 16-ton skip, a trip must be made every 38 sec. to obtain an output of 1500 tons per hour. I believe that a skip can be designed to dump 16 tons in 10 sec., but allowing 12 sec. to dump the running time per trip will be 26 sec. Assuming 5 sec. for acceleration and 5 sec. for retardation, the maximum rope speed will be only 1925 ft. per min., and it will not be necessary to slow down through the dumping horns. The

maximum power is practically the same at three points of the cycle at 2500 hp., and there is no negative power during retardation. The two cycles are illustrated in Fig. 9. The cycle for the 16-ton skip indicates that the accelerating and retarding time could be easily lowered to 4 sec., which in turn would reduce the maximum rope speed to under 1900 ft. per min.

The capacity of the hoist motor would be 1800 hp. against 4000 for the 11-ton skip; the generator 1500 kw. against 3300 for the 11-ton skip; and the a.c. motor 1600 hp. against 1900 for the 11-ton skip. The flywheel would weigh about 60,000 lb. instead of 90,000 lb. for complete equalization with a 10 per cent. slip. The power required for hoisting each ton of coal would be 0.95 kw.-hr. per ton for the 11-ton skip and 0.8 kw.-hr. per ton for the 16-ton skip; this represents a saving of about 16 per cent. in power.

A straight cylindrical drum for the 16-ton skip would require a 2000-hp. motor, but this motor would be 25 per cent. higher speed than the 1800-hp. motor, if we use a 12-ft. drum, so that it would theoretically cost less. The actual power required would be about 2 per cent. more, because of the greater losses during the accelerating period.

The foregoing are two important points that should be given serious thought when considering future hoists of the same magnitude.

George S. Rice, Washington, D. C.—About a quarter of a century ago when I was connected with various mining operations in Illinois I designed a number of the first steel head-frames and tipples installed in the West. At that time a capacity of 2500 tons in 8 hr. was considered to be the maximum tonnage advisable to plan for a mine capacity. As the maximum has now been pushed up five times that of 25 years ago, what may we expect in the next 25 years?

Eli T. Conner, Scranton, Pa.—Are the rooms driven on 40-ft. centers because you contemplate recovering?

John A. Garcia.—There is no standard or fixed method used for drawing the pillar; 18-ft. ribs are left between the rooms.

Eli T. Conner.—The rooms are driven how wide, 24 ft. with 16-ft. pillars?

John A. Garcia.—Rooms are 22 ft. wide and pillars 18 ft. wide.

George B. Harrington.—We contemplate drawing the pillars. We have not driven to the boundary except in one or two places where the boundary is close to the shaft.

We have quite a few plans for trying to develop a system to handle the mechanical loading but everything is experimental. We plan shortly to be able to start the retreating system. We have had to go ahead and get tonnage on the advancing method without waiting to retreat, without

waiting until we could get enough development to come back and pull the pillars.

ELI T. CONNER.—What success have you had in recovering the ribs where first mining has been done with 24-ft. rooms and 16-ft. pillars?

GEORGE B. HARRINGTON.—We have had moderate success in some of the other mines. The first year we won't get the efficient recovery that we are planning for, but the design of the mine has been made such that high recovery is expected as the cross entries are spaced so that by reducing the number of rooms we will have sufficient pillars left so that good recovery will be obtained.

Room will probably be 80-ft. centers and not driven more than 20 ft. wide on the advance. With this method, I believe we should get a recovery of better than 80 per cent. of the entire coal bed.

F. F. JORGENSEN, Gillespie, Ill.—What effect has the falling of 10 or 12 ft. of coal on the surface?

GEORGE B. HARRINGTON.—The company owns 4000 acres of surface over this piece of coal and where the surface is not owned by us, it is rolling and subsidence has not caused trouble. The mine is 500 ft. deep and while the 10 ft. thickness of coal is large, the amount of subsidence is very small. Where our mining operations are under the state highway, railroad bridges or other important structures, we have left enough pillar to assure no damage will be done by subsidence.

F. F. JORGENSEN.—Are the rooms and pillars in the old mine the same as in the new?

GEORGE B. HARRINGTON.—Approximately the same. We are trying to develop a better scheme so that we can use the loading machines to advantage. They have been worked in the rooms designed for hand loading and it is quite a problem to rearrange the mining system in order to accommodate the loading machine.

F. F. JORGENSEN.—The 8-ft. pillars do not crush?

GEORGE B. HARRINGTON.—We have had some crushing of pillars at No. 1 Orient in panels that were about to be abandoned.

F. F. JORGENSEN.—When we use less than 25-ft. pillars they crush immediately, so that we have to change to 30-ft. pillars. We have a limestone cover—30 ft. of cover lying directly over the coal—which may have some bearing on the matter. What is the condition of your mine?

GEORGE B. HARRINGTON.—There is no rock immediately overlying the coal; the first 400 ft. of cover over the coal is shale. Over this shale and about 80 ft. below the surface of the ground there is a bed of sandstone which runs from 10 to 30 ft. in thickness.

F. F. Jorgensen.—In regard to Mr. Bright's computation on using larger cars and the consequent decrease in the consumption of power, was the fact that there is a limit in the size of mining cars taken into consideration?

Graham Bright.—At present, mine cars of 8 to 10 tons are being built. With mechanical loading and high coal, still larger cars could be and probably will be built.

F. F. Jorgensen.—With the use of the larger car, there is a great increase in the degradation of the coal.

Graham Bright.—The larger the car the less is the degradation; this is found to be the general practice in many places.

E. T. Gott,* Pittsburgh, Pa. (written discussion).—Our contract was for the two complete openings from coping to sump, including the work in the bin station at the bottom of the main shaft and the usual construction in the coal at the auxiliary opening. From the nature of the surface material shown in the bore-hole records for the West Frankfort district, it was expected that the caissons, after penetrating about 43 ft. of top clay and loam, would have about 32 ft. of running sand to contend with before striking the shale, shown in many of the holes at 75 ft. below the surface. Experience, though, indicated that this sand contained very little water; so while every preparation was made for sealing under air, we were confident that the two shells could be safely landed by open methods, and the contract so read.

Up to that time, many shafts had been lost by attempting to sink through the water-bearing sand with skin timbers and jacking plates. With the thought that we might have to resort to air to affect a seal, the concrete deck and working chamber were formed in each caisson and the three 7½-ft. digging tubes placed, ready for the reducing hoods and connections to receive the man and material locks, should they be needed.

At the main shaft, the dimensions in the clear of the concrete lining in rock were 9 ft. by 27 ft. 10½ in., with straight sides (24 ft. tangent) and rounded ends. To allow for irregularities in sinking, the caisson was, therefore, made 10 ft. 6 in. by 31 ft. 6 in. The design showed a wall thickness of 3 ft., giving a 457-lb. per superficial square foot allowance for skin friction, which we knew to be ample for the material through which we were to pass. The steel shoe was of angle and plate construction, and the concrete (of 1:2:4 mixture) was heavily reenforced.

In the 10 ft. of excavation to the point at which the shoe was to be leveled-up, all sloping and trimming were done by hand and the excavation removed by orange-peel bucket. Clearance is given at the toe of the

* Vice-president of the Dravo Contracting Co.

slope for setting and riveting the segmental sections of the shoe. This entire framework is supported by short lengths of 2-in. plank, so placed under the cutting edge that they will work to the inside when this first lift of the caisson is lowered; these details are shown in Fig. 10.

With so little lateral support, there is a natural tendency for the caisson to list when passing through surface loam; so this preliminary excavation is made and the slopes used for bracing and guiding the shell in its first downward movement. With this 12 ft. lowered and the second form added, the movement is controlled at the shoe by undercutting the high side or retarding the cutting edge at the low side, as may be necessary. In Fig. 3, about 30 ft. of concrete had been placed and lowered and the

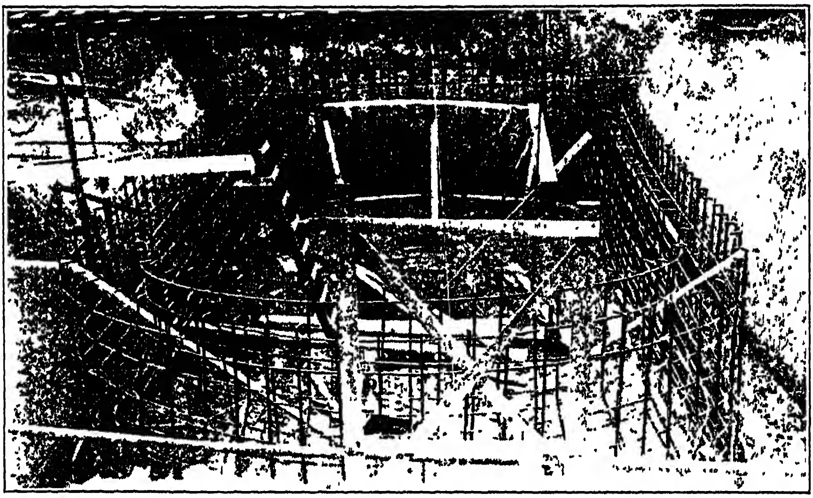

FIG. 10.—CONSTRUCTION AND REENFORCEMENT OF CAISSON.

outside form moved up for the next lift. With the placing of the horizontal reenforcing and the setting and lining of the inside form, the shaft was again ready for concrete. The three 7½ ft. digging tubes are shown in place ready for the attachment of the reducing pieces to the air locks, in case of failure from the open method.

Work at this shaft was begun Apr. 13, 1921, but for three weeks non-delivery of reenforcing steel and other things greatly delayed the work, so that the first form was not poured until June 2. After allowing five days for the concrete to set, the form was carefully lowered by hand to insure an even movement and to reduce the stresses in the comparatively green concrete. Alternate excavation with orange-peel and concrete additions then followed until the sand was reached at 50 ft.

At the top of the sand, with the orange-peel digging in the blind, the excavation was carried 7 ft. below the level of the shoe, without any

movement being recorded, or a realization of the condition on the bottom. When the lubricating pipes were put into play, the shell suddenly dropped 8 ft., and vertical cracks developed in both walls, about at the cross-center line. The inadequate timbering, then in place, was removed and 12 by 12 in. timber sets on 3 ft. centers were wedged into place, as shown in Fig. 11. The caisson was then 63 ft. below the surface, and the sudden drop was into the sand. The orange-peel bucket would not handle this fine-grained material so a tightly-closing clamshell was used.

Two months after the first form was poured, we were down 71½ ft. with 83½ ft. of concrete in the shell and the cutting edge so far into the sand that its movement could not be controlled by off-center digging with

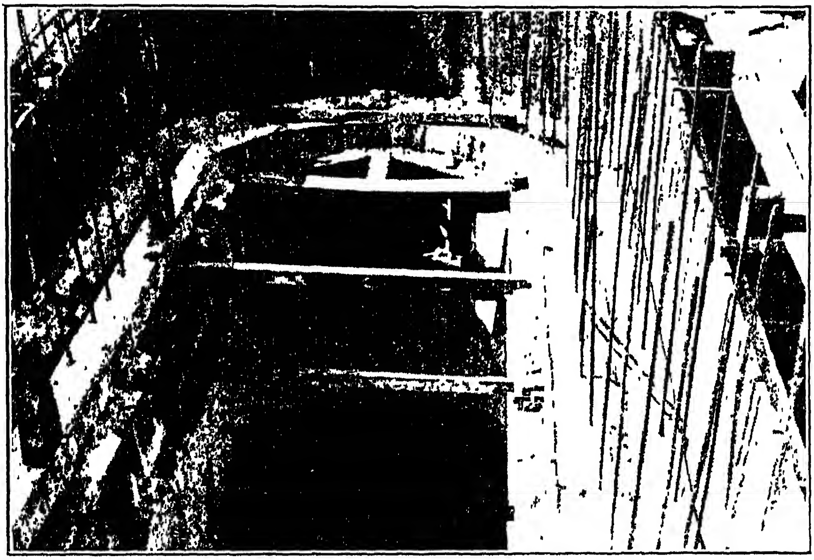

FIG. 11.—CAISSON WITH OUTSIDE FORM MOVED UP READY FOR NEXT LIFT, DIGGING TUBES ARE SHOWN IN PLACE.

the clamshell. So little resistance was offered by this material that we had to do our plumbing from the surface. On Sept. 1 the shell was at 83 ft. but was 19 in. out of plumb, with its top to the east.

Obtaining no settlement after two days' digging, and noticing, from the nature of the material brought up, the closeness of the shoe to the rock, it was decided to pump out and undercut the shoe. This dewatering of the hole was accompanied by many "runs" of sand, with much time lost in sand-bagging and in pump troubles from gritty water.

After a few shifts on the bottom, the shoe was lowered to a point just above the rock on the east side. This being the low side, it gave us an opportunity for blocking the cutting edge from the solid. The top west was then heavily loaded with rails and a carload of 6-in. H-beams, but

no movement could be obtained and there was no tendency for the shell to straighten itself. We then removed the load at the top and had a driller put down 24-in. core holes to the west of the shaft to unbalance, in this direction, the earth pressure on the two sides of the shell, but heavy rains, after four holes had been bored to 50-ft. depths, caused the holes to cave-in, so the idea was abandoned. The drill outfit was removed and the 100-ton cantilevered load replaced. We then resorted to outside jetting with water, steam, and air, using long pipes from the surface in conjunction with the lubricating effect from the system of 2-in. pipes at

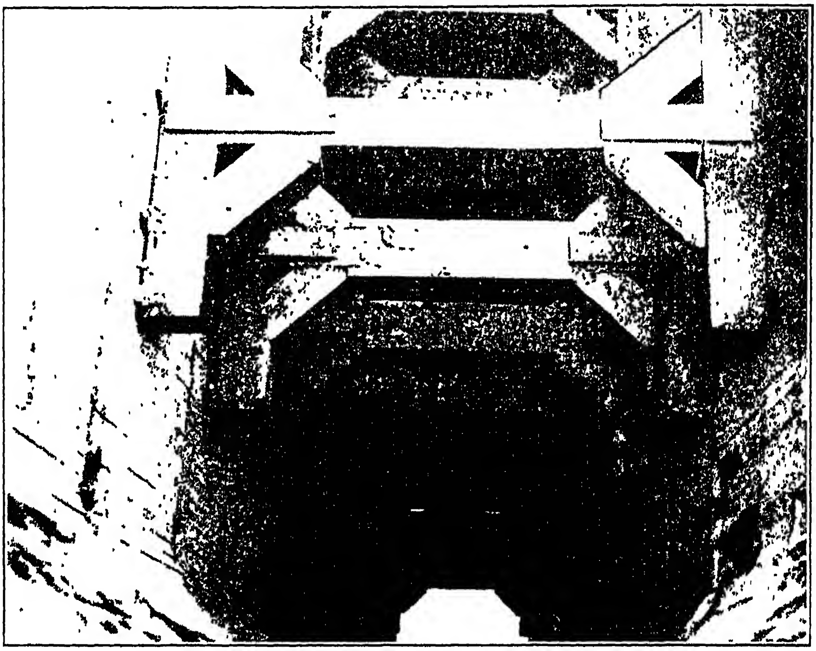

FIG. 12.—BRACING MAIN SHAFT.

regular intervals in the caisson walls. The caisson seemed to respond to this treatment and, noticing a slight movement, we again pumped down and started clearing out the working chamber. Mucking under the west cutting edge, checking the east side with short inclined greased rails, and outside jetting along the west wall, continued for two weeks; a check then showed that the shell was 13½ in. out of plumb, with the shoe resting solidly on the rails on the east side and the entire side opposite was undercut to the rock and unsupported. So in a final effort to throw the top of the shell in this direction, 30 ft. vertical was gouged out, exposing the entire upper half of the caisson. The effect of the inclined rail under the east shoe tended to throw the bottom in this direction and. with the load on the top acting, the shell soon moved over until it was only 5½ in. out of plumb. The top loading was then removed and the final adjustment

made with the hole pumped out and men inside. Fig. 13 gives an idea of the extent of this outside digging.

On Oct. 19, 1921, the regular rock shifts were formed with the shoe sealed at the rock and the shell as plumb as it could be gotten, and well within the 9 in. allowance for contingencies of this sort. The cutting edge had been landed 87½ ft. below the surface and had been carried into the rock 5 ft. before it could be plumbed and suitable material found for sealing.

The main shaft is designed for 13-ton skips; in addition to these two skipways, there is a compartment for a steel stairway at one end of the shaft beyond the buntons and a small area at the other end for pipes and wires. The three buntons are 6-in. 24.1-lb. H-beams, with their ends

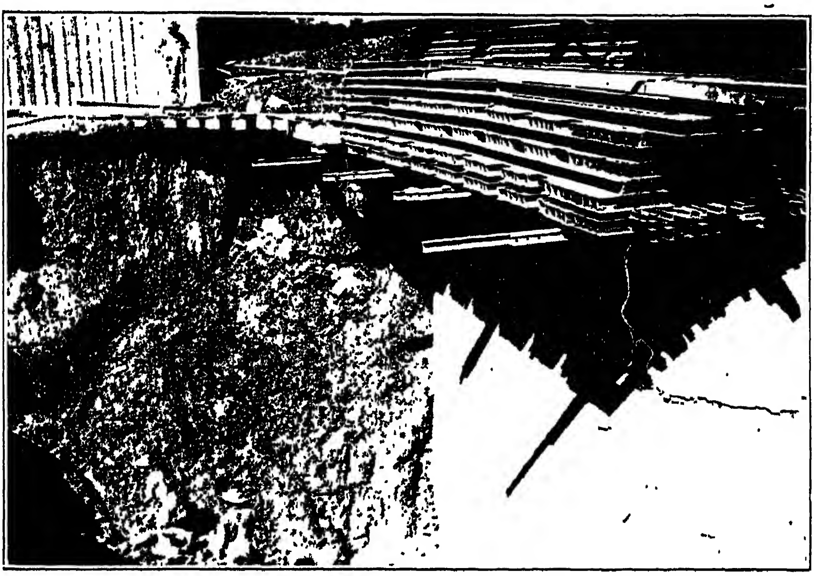

Fig. 13.—West side of main shaft, loaded and excavated for 40 ft.

grouted securely into hitch pockets and placed with their flanges vertical; the pocket so formed was filled with concrete. The four 80-lb. rail guides were in 30-ft. lengths, with splice plates ground to fit the railhead. The steel stairway was of Allen & Garcia design, using diamond-bar treads and platforms, and extending from the coping to the bottom landing at the bin platform. The excavation in rock was carried 27½ ft. below the shoe and the first 36 ft. run of concrete lapped 8½ ft. on to the caisson, forming a good bond between the roughened caisson walls and the lining then poured, and effecting an absolutely watertight seal at the closure. Sinking and lining in rock, at 40 ft. intervals, continued without serious interruption to the point below the coal from which the bin excavation was started.

It was then necessary to drive 500 ft. from the shaft in each direction to test the coal and to determine the proper elevation at which the rotary dump should be placed; when this was obtained it was possible for us to tie-in the entire bin arrangement. Six weeks, therefore, were spent after we reached the coal in this entry driving, followed by 1824 cu. yd. of excavation actually removed for this bin chamber and the pouring of 691 cu. yd. of concrete. With the bin completed, the three lines of H-beam buntons were placed, lined-up and grouted, and the four runs of 80-lb. rail guides added.

The placing of the steel stairway, from coping to coal, was the last work at this opening. After mucking-out the debris that had accumulated in the sump, this shaft was ready for acceptance by the coal company on Oct. 9, 1922.

To strengthen the caisson walls at the cracks that developed with its sudden drop at the top of sand, it was decided to brace the walls with additional H beams along the lines of the regular buntons. These beams, on 3-ft. centers, were placed with their flanges vertical, and provided with 12 by 12 by ¾ in. sole plates riveted to each end and brought to an even bearing against the walls of the caisson by means of steel wedges.

This framework extended from the top of the deck, about 7½ in. above the shoe, to the coping, and, with all beams wedged into place and the 12 by 12 in. timber sets removed, a 9-in. facing of heavily reenforced concrete was poured, utilizing the allowance that had been made in the original design for caisson irregularities, and flushing the new lining with the neat dimensions in rock.

As an additional precaution, forms were placed along the two lines of buntons at the center of the shaft (they were thought unnecessary at the end bunton) and two 6-in. panels of reenforced concrete were poured monolithically with this inner lining and for its full depth. This gave the entire job a finished appearance and the result was most satisfactory.

For the auxiliary shaft, the design called for two cage compartments for men and materials, a small area at one end beyond the buntons for pipes and wires, and an air compartment at the other end beyond the cage ways. This 130 sq. ft. airway was confined to this end of the shaft by a 6-in. reenforced-concrete curtain wall along the line of the end buntons. The inside dimensions of the shaft in rock were 13 by 31½ ft. and, as at the main opening, an 18-in. allowance was made in the caisson width for irregularities, giving 14 ft. 6 in. by 33 ft. 1 in. for the neat inside dimensions of this shell. With a 3 ft. wall thickness, the allowance for skin friction was 464 lb. per superficial square foot, with other details corresponding with the design used at the other opening.

Work was started June 2, 1921, seven weeks after ground was broken at the main shaft. As before, the preliminary excavation was made to the 10-ft. depth, the shoe set and riveted, and the working chamber and

air deck formed and poured; 13 ft. 9 in. were lowered in the first operation, and alternate sinking and concreting carried the shell to the sand at 67½ ft. without any serious delay. Hair cracks were noticed when the caisson dropped into the sand, and the 12 by 12 in. timbers were placed from deck to coping. We moved uniformly through the sand and, three months after ground was broken, the shoe was at the rock at 81 ft. 5 in. and practically plumb. At this point, the shell became light and the top was loaded with rails, H beams, and sand bags, which carried the cutting edge through the shale and landed with an even bearing at 82 ft. 10 in. below the surface. In this entire depth, the shell was 1 in. out of plumb and, with the addition to the coping to bring it to plan elevation, the caisson measured 86 ft. 9 inches.

In rock, the regular sinking shifts were formed Sept. 9, 1921, and the first run of lining lapped 10 ft. on to the caisson and afforded a perfect seal. A fine sinking average was maintained, until the bottom of coal was reached at 488 ft., when it was again necessary to drive 500 ft. from the shaft to establish the proper elevation for the bottom landing. The H-beam buntons were then added, and the three runs of beams were lined up for 460 ft., followed by the pouring of the 6 in. curtain wall to this same depth and the placing of the four runs of 6 by 10 in. timber guides. This work was done while waiting for the determination of grades at the main shaft. When this information was obtained, the bottom work beyond the shaft was begun. The buntons, guides, and curtain wall were then extended to the coal and the shaft was turned over to the coal company on Aug. 11, 1922.

When the shaft was given its final inspection, the lining was found to be so irregular, because of indifferent form setting by the men in the hole, that the full clearance called for could not be gotten. The coal company agreed to cut down their cages to suit. A statement from the company in October, 1924, showed that the trouble at both shafts had been overcome, with a full clearance at the Auxiliary Shaft and sufficiently trimmed at the other opening to allow their skips to operate safely.

Under the agreement with the union under which we worked at West Frankfort, the carpenters were not allowed to go into the hole to set forms. The accuracy with which this part of the work was done, therefore, depended on unskilled crews of shaft men. In every case, the first form had been accurately set but, as the succeeding 6-ft. lifts were placed, they gradually got out of line, so that the form had to be distorted to make the proper closure at the old lining above. It was at these correcting points near the closures that the trimming had to be done. It is unusual, in concrete lined shafts, for the form to vary more than ½ in.

The job was completely unionized and, at the start, was run entirely to the liking of the men who formed the first shifts. There were, naturally, many claims by them of infractions of the rules of their local and

several strikes occurred as a result. These walkouts caused much lost time and being, in nearly every case, decided in our favor created an ill-feeling. The worst element was, therefore, working against us and not for us; but, taken as a whole, the shaft men were capable and, after becoming familiar with our methods, became very efficient. The best month's work was done in March, 1922, when 114 ft. of sinking and 117 ft. of lining were completed at the auxiliary shaft, with 106 ft. of sinking and 114 ft. of concrete at the main shaft. The following units were regarded as final for the job:

Hoist caisson from surface to solid rock	87.5
Hoist in rock to bottom of sump	459.67
Total depth	547.17
Auxiliary caisson to solid rock	86.7
Auxiliary in rock	335.0
Total depth	421.7

George N. Simpson,* Chicago, Ill. (written discussion).—It is only a few years since the American Coal Mining Co. of Bicknell, Ind., claimed the world's record at approximately 5000 tons with a shaft equipment that was designed and expected to deliver a capacity of approximately 2000 tons. Shortly afterwards, Superior No. 3 at Gillespie, Ill., took the world's record, and it was held by several mines at different times with capacities somewhat over 5000 tons in an 8-hr. shift. A few years ago, however, the Orient No. 1 mine of the Chicago, Wilmington & Franklin Coal Co. broke all previous records by hoisting over 6000 tons in a day, and since then has held the record with ever-increasing capacity up to 8218 tons.

All these mines were designed for a capacity not to exceed 2500 or 3000 tons in 8 hr., and it was only by enormous efforts and at the expense of costly and constant repairs, and also at the expense of safety, that these large tonnages could be obtained.

. The writer has visited a number of these mines and has been on the shaft bottom when the mine was hoisted in excess of 5000 tons in a shift. It is not unusual to find on the shaft bottom the highest paid officials of the mine, using every effort, both mental and physical, to cage the cars and get the coal to the surface as fast as it can be loaded in the workings. A visit to the shaft bottom of the New Orient mine, or any one mine equipped with similar hoisting machinery, is quite a contrast; orderly routine and the apparent ease of operation make it practically impossible for a casual observer to realize that probably twice the tonnage is being handled as at the old type of mine with all of its hurry and excitement.

* General Manager, Car-Dumper & Equipment Co.

About 18 years ago, Joseph Leiter decided that there was no reason why the Bell & Zoller mine could not be designed and built to handle a capacity of at least 7000 tons in a day, and it was so designed and equipped. But through lack of knowledge of the requirements for handling such a large tonnage and the lack of suitable machinery to handle the coal at the shaft bottom, the operation of the mine, as far as the hoisting of the coal, was rather unsuccessful. About 7 years ago, however, at the Bell & Zoller No. 1 mine the hoisting capacity was increased, and equipment was installed to deliver coal to the skips as rapidly as it could be hoisted. Of course, it took some years for the mine to be developed so as to produce a tonnage which would keep its hoisting equipment busy, but year after year the capacity of the No. 1 Bell & Zoller mine has been increasing until at present it holds undisputed claim on the world's record. As a result of the success at this mine, several other mines were equipped with similar machinery.

The maintenance of a required capacity of hoisting with the old type of cage hoist was a serious problem in the development and the output of the mine was practically limited by this part of the equipment. When we realize the large amount of money represented in the investment in one of these shaft mines, it is not economical to permit the capacity of the mine to be limited by the ability to hoist the coal and deliver it to the railroad cars; especially where the cost of the hoisting equipment and car unloading equipment represents such a small proportion of the development cost of the mine. Also, safety is not only the most humane policy but the most economical policy; therefore, the unsafe condition developed by overworking the cage-hoisting machinery as well as the frequent failure due to overtaxing, had caused a great many of the leading engineers to decide that larger capacity hoisting equipment must be developed to take care of the tonnages of these larger shaft mines.

The New Orient mine, at the present time rated at a 12,000 ton capacity, is by far the largest capacity shaft installation attempted. If it is posible to get the coal to the shaft bottom, I do not doubt that its rated capacity will be greatly exceeded as underground loading machinery and haulage equipment are improved. But the hoisting equipment and car handling equipment have been generously designed, so that it will be many years before they can be overloaded.

Another big advantage is that an installation of the type found at this mine permits the use of solid-body dust-tight, water-tight cars. This is important in a mine that has any tendency to be dry or that is at all gassy. It is generally conceded that there must be at least 55 per cent. inert matter in coal dust to make it non-combustible, and the Rocky Mountain Coal Mining Institute recommends 80 per cent. inert matter be maintained where the dust is very fine. This means that for every 20 lb. of fine coal dust that leaks around the end gates or through the bottoms of

the mine cars, the operator must add 80 lb. of rock dust to make his haulageways safe.

Up to the introduction of skip hoisting in shaft mines, the size and shape of the mine were largely determined by the size of the shaft; but skip hoisting permits the cars to be designed at a maximum length and width permitted by the mining conditions.

But it requires more than hoisting and car-unloading equipment to make a big mine. The Franklin County seam in southern Illinois and a few other seams in other parts of the country are favorable to the development of very large coal mines. In other localities, where the coal is not so thick, where the grades are heavy and haulage conditions severe, no type of hoisting equipment would insure the development of a large mine and the maintenance of a large capacity. The development of loading machinery, which is being given so much attention at present, and the further development of haulage equipment may make it possible to increase considerably the capacity of some mines that have only a moderate output. Money may be wasted in the installation of large hoisting equipment if all conditions are not carefully studied.

W. M. Hoen, Chicago, Ill. (written discussion).—Skip hoists, while common in metal mines, are unusual in coal mines, and in this mine there are very severe hoisting conditions. In some coal mines, there are steam hoists operating at very high rates of acceleration and fast cycles; but nearly always these have platform cages and small cars. But here the load is 13 tons and the skip weighs 17,000 lb., which with the 2-in. ropes and the 17-ft. drum, using the high acceleration rates necessary, make the inertia power requirements very large.

The mining requirements call for 138.4 trips per hour, or 2.3 trips per minute; 9 sec. is specified as the time necessary to dump the coal, which only leaves 17 sec. actual running time to travel 607 ft.; 5 sec. is allowed to accelerate the motor from rest to a full speed at 75.5 r.p.m. This gives a rate of acceleration of 13½ ft. per sec. squared for the descending skip, which is close to the practical limit.

The torque requirements during acceleration are equal to a 7000 brake-horsepower input to the hoist. This is provided by the two coupled motors, which are rated at 2000 hp., each on a continuous-duty basis. The positive ventilation allows the motor armatures to be somewhat smaller with a great decrease in the inertia of the rotating parts, and a corresponding decrease in the power input during acceleration.

During the period of full-speed running, the rope speed reaches 4030 ft. per min. This allows 6 sec. to bring the skip to rest, and the rope speed must be reduced to 900 ft. per min., which is the maximum speed specified for the skip to enter the dump horns; therefore, the rope speed must be reduced from 4030 to 900 ft. per min. in 3 sec. This high retarding rate requires a braking torque of 3800 hp. This energy must be absorbed

by the hoist motors acting as generators, the energy being chiefly stored in the 90,000 lb. flywheel on the motor-generator set.

The motor-generator set presents no unusual features. The large power inputs into the hoist are equalized by means of this set with the flywheel, and the necessary control equipment so that the power requirements from the line do not exceed the capacity of the 2200-hp. driving motor.

The control is of the contactor type, being full-automatic for acceleration and also during retarding period. This type permits introduction of all necessary safety devices, such as overtravel and overspeed; also, the slow-down can be controlled in sections. Speed below maximum limit is under the control of the operator, who handles a small drum controller through the usual control lever. The automatic features permit the acceleration to be maintained at a maximum even rate; otherwise this fast cycle could not be maintained safely. This type of equipment lends itself to this application; the hoist operates so smoothly that it is only by reading the meters that one realizes the large forces that are in action.

One unusual feature in the installation is using washed air for cooling the motors. High-pressure oil supplied by small pump, at approximately 1500 lb., lubricates the lower half of the wheel bearings just before starting the set; this results in greatly reduced initial power input. All bearings are lubricated by oil rings, with the addition of oil-circulating system, and the motor-generator set bearings have water circulation as a matter of safety.

As an additional safety feature for emergencies, dynamic braking direct on hoist motors is provided, by automatically introducing a heavy capacity resistance directly across the motor armatures.

To provide for quick stopping of the set, auxiliary switches and resistances are provided for reversing the motor of the motor-generator set, the power input being limited to the full-load rating. Under this condition, it takes about 7 min. for the set to come to rest; without this provision, it would take over an hour.

In the installation of this equipment a full basement has been provided, which permits a bus system of wiring in place on the usual cables. This makes a very neat and workmanlike job and provides access to all equipment.

R. W. McNeill,* East Pittsburgh, Pa. (written discussion).—The results at New Orient are unusual not on account of the use of unusual tools, but on account of an unusual use of the tools at hand. These tools included modern coal-conditioning equipment, central-station power, the electric hoist, the electric mining locomotive, automatic substation equipment, and mechanical loaders. Conditions in the bituminous-coal

* General Engineer, Westinghouse Elec. & Mfg. Co.

industry are such that only well-equipped, well-operated mines can meet the competitive conditions. Central-station power has come to stay. The coal operators have found that it is more economical to purchase their power than to produce it.

The outstanding features of the New Orient mine are the electric-hoist installation and the automatic substation installation for furnishing power to the mining locomotives. The engineers of the C. W. & F. Coal Co. have installed not only the largest electrically driven coal-mine hoist, in point of rating of electrical equipment but also in point of capacity to handle coal. This hoist has many unusual features: it is the first large coal-mine hoist to use two direct-connected motors for its operation, it operates at the highest maximum rope speed used for an electrically driven coal hoist and power for operation is furnished by a flywheel motor-generator set of much greater capacity than has been used heretofore for the operation of mine hoists by the Ilgner Ward-Leonard system.

The automatic substation installation, while not quite so spectacular as the main hoist, is as much an achievement. By the use of automatic substation equipment, the engineers of the company have been able to distribute power for the locomotive haulage system in an economical and efficient manner not only to take care of their present requirements but also to assure the same efficiency and economy for future extensions.

The experimental work toward the development of mechanical loading of pit cars will prove of great benefit to the industry and it is practically certain that mechanical loading will be the common practice in coal mines in the not distant future. Progressive coal operators working in conjunction with the builders of coal-handling machines are rapidly solving this problem and it will be only a few years before the mechanical loader will be as necessary to the economical production of coal as the electric haulage motor is today.

Chas. C. Whaley,* Knoxville, Tenn. (written discussion).—It took considerable courage, back in 1922, to put loading machines in an Illinois mine. Most coal operators of that state said that the experiment was doomed to failure, for they believed that organized labor would place so many restrictions on the use of the machines that they could not be an economic success. In spite of this the author purchased loading machines for development of the mine, the first loading machine being a No. 4 Myers-Whaley, which was put into operation less than 100 ft. from the bottom on Dec. 26, 1922. Later, additional Myers-Whaleys and a number of Joy machines were put in and the entire development work of this mine was done mechanically.

Carl Scholz,† Charleston, W. Va. (written discussion).—The Orient hoist is by far the largest ever designed for a coal mine and the query

* Sales Manager, Myers-Whaley Co.

† Vice-president, Raleigh-Wyoming Coal Co.

arises whether a conical-drum equipment produces the best results under the existing conditions, because the primary object of skip loading is to reduce rope speed to a minimum; and a 17-ft. drum running at 75 r.p.m. produces a rope speed of nearly 4000 ft. which defeats the object of skip hoisting because it nearly approaches the maximum speed for deep shaft hoisting and seems far in excess of what would be considered good practice for a 700-ft. lift.

One feature prompts me to suggest that a straight-face drum would have been preferable; namely, that it frequently becomes necessary to stop the ascending load just before the skip enters the dump horns, at which time the ascending load is on the maximum diameter of the drum and the descending load or empty skip is on the smallest diameter, thus causing quite a peak when the load is started up again. As no reference is made to a storage bin in the main tipple, it is assumed that only the ordinary hopper feeder, holding perhaps twice the contents of the skip, is available, and in regular operations, therefore, frequently the hoist has to stop while the feed hopper is being emptied. It is true that the flywheel set will absorb this peak, but once the flywheel set is decided on, a conical drum not only unnecessarily increases the first cost but also the power requirements, upkeep expenses and rope wear. If depths of 2000 ft. and over were to be considered, my argument would not apply because the difference in the ropes would have a material effect on the hoisting load, which is not the case with the comparatively shallow hoist of 700 ft. Conical drums applied to electrical hoists seem to have a place more where smaller loads are employed or where no stops occur after the hoist has been belled away. The shaft and drum of conical equipment is longer, hence heavier and more costly. In this connection I call attention to the hoist at the Valier Coal Co., where a single motor hoist with a 1350-hp. motor and a 700-ft. lift has handled 8664 tons without undue heating on a 9-ft. straight-face drum.

The loading of skips is an interesting operation because only 9 sec. are given to the loading of the skips, which means a flow of nearly 2 tons of coal per second. With a volume of 12,000 tons per day to be handled, it is most essential that the skip, on each trip, carry a full load; and where the contents of individual cars are dumped this cannot always be obtained, because it is rarely ever feasible to load cars to their full carrying capacity, particularly when working faces are being cleaned up. The solution to this question seems to lie in the construction of a measuring hopper located below the dump, in order to assure a full load to each skip. This hopper would be filled while the skips are traveling, but it involves one additional unit of machinery and one additional movement of the coal. The latter movement may be objected to as increasing the breakage of coal, but it is now generally conceded that hoisting by skips

results in no more breakage of coal than hoisting in cars and handling by self-dumping cages.

While the distances in Orient No. 2 will not be great, and power losses, due to voltage drop, will be small, economies would have been effected by having the motors of coal-cutting machines, coal-loading machines, compressors, etc., equipped with alternating-current motors, 220-volt, thereby saving considerable copper and investment in motor-generator sets. In gaseous and dusty mines, the absence of sparking of commutators is a feature deserving consideration.

Systems of Coal Mining in Western Washington*

By Simon H. Ash, Carbonado, Wash.

(New York Meeting, February, 1925)

The coal-mining districts of Washington are mainly west of the Cascade Mountains; Fig. 1. The mines are on the foot hills of the slope, the lignite fields of Lewis and Thurston counties extending into the valleys west of the mountains. An exception is the Roslyn field, a small but important area on the eastern slope.

If western Washington is divided into three areas from north to south, the bituminous coal fields and most of the active operations will be found largely in the middle area. The southern portion contains lignites over a large area. In the northern portion, mining is practically restricted to one mine, the Bellingham, which is working a subbituminous coal seam near the city of Bellingham.

The degree of alteration that the coal of a particular field has undergone may be gaged roughly by the position of the field with reference to the Cascade Mountains. The lignites of Tenino, Tono, Mendota, and Castle Rock, on the railway line connecting Seattle and Portland, occur in a region of low relief, in which the Eocene coal measures have suffered only minor disturbances. Subbituminous coals occur in the foot hills, as at Renton, Newcastle, and Bellingham. Coals high in fixed carbon are found nearer the mountains, where the measures have been sharply tilted and folded, as at Black Diamond and Carbonado. The two typical anthracite fields are located still higher, in the rugged mountains where outcrops appear both in deep gorges and on ridges at elevations approaching 5000 ft. At Carbonado is found the only semianthracite seam worked on a commercial basis. The occurrence of this seam in a region of bituminous coal is due to the nearness of an igneous sill, which has changed its character from bituminous to semianthracite.

Purpose of Paper

On account of the various faults and dips and the varying nature of the wall rock, many difficulties are encountered in mining and winning the coal. The paper gives some of the methods used in mining and, in some detail, the conditions that affect the efficiency of these methods.

* A thesis presented at the College of Mines, University of Washington, in 1924.

The paper does not discuss the geology of the district, except incidentally, as numerous reports have been issued on the geology; nor does it attempt to discuss the methods of preparation of the coal for market, although

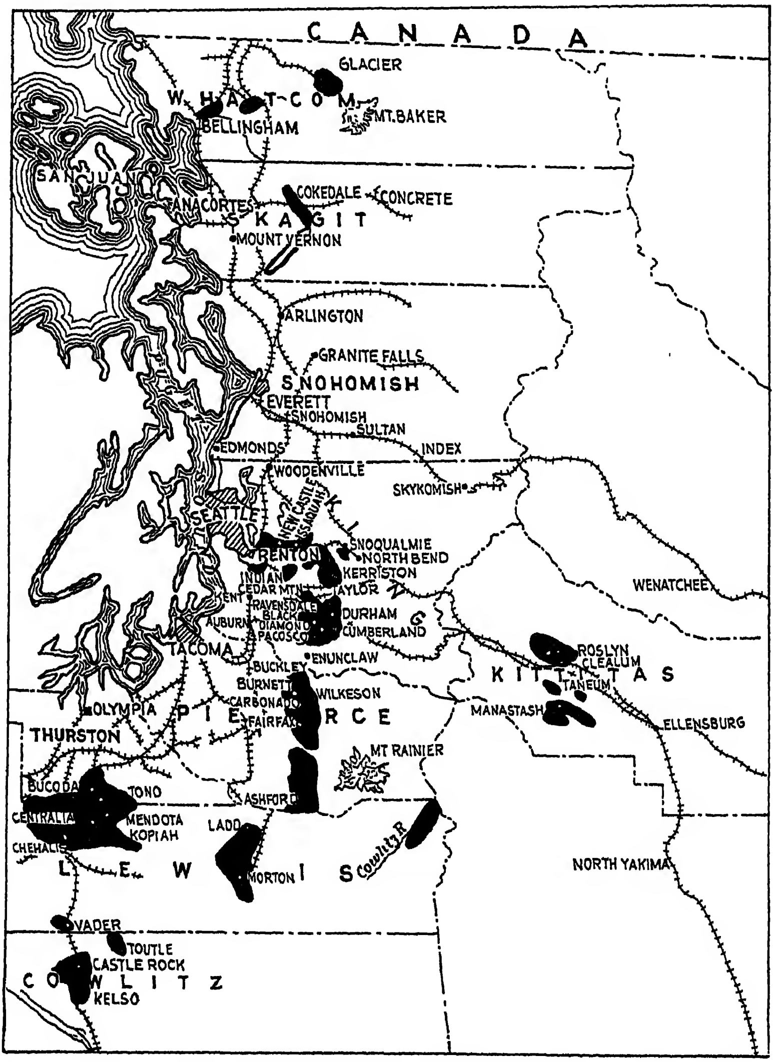

FIG. 1.—COAL AREAS OF WASHINGTON.

this is an important part of the operations. Details of underground methods that have been tried and found successful are given as well as those that have not proved successful.

GENERAL GEOLOGY

The coal-bearing rocks of Washington are of Eocene age and correspond, in time of deposition, with the lignites of the Dakotas, Montana,

and Texas. In Washington, however, mountain-building forces were active in late Tertiary time, faulting took place on a large scale, intrusive agencies were active in some fields, and much of the coal was changed in rank and structure. In general, the surface has a rugged contour in the different districts. The covering of glacial material masks the continuity and only a few of the fields are well marked. Because of this, the fields have generally been subdivided arbitrarily, according to the county in which they occur; and reports on the coal areas of the state have been differentiated in this manner.

All coals contain foreign material in the form of partings or binders, but the coal of Washington includes more of such partings than the coal of many other parts of the United States. The old swamps in which the coal was deposited were apparently subject to many floods which washed in mud and sand from the surrounding higher lands. These materials formed a shale, sandstone layer or parting in the coal bed, which detracts greatly from its value, for the partings must be removed before the coal can be marketed. Removal is generally difficult and expensive. This foreign material cannot be gobbed in the mine, on account of the heavy dips, but must be handled with the coal and eliminated at the cleaning plant.

Mining Methods

A study of the various factors involved in mining coal in the different counties shows that the state can be divided into two divisions based on the methods of mining. These two divisions are Kittitas, Thurston, Lewis, and Whatcom counties, which will be designated division 1, and King, Pierce, and Skagit counties, with a future development in Whatcom, called division 2. The production of the two divisions is approximately equal under normal conditions. The mining methods in division 1 do not present any problems that might be classed as unusual. They do not vary from those in many parts of the country and are not discussed here. As a whole they are flat workings, the coal seams are clean, the rock can be separated at the face and gobbed, and the roof is good. The workings of division 2 are on the high-dip seams of the state, where everything between walls must be removed in mining and the rock and accompanying refuse removed in cleaning plants on the surface. The walls, as a rule, are bad and the methods of working are diversified. The daily and yearly output per man in division 1 is greater than that of division 2; the payment of miners in division 1 is on the tonnage basis, while in division 2 it is mostly on the contract yardage basis. Practically all the coal in division 1 goes directly from the mine cars into the railroad cars, while all that of division 2 must be washed or prepared for the market. The coal of division 1 is used mainly by the railroads, while that of division 2 finds its market as a domestic

fuel, for coke, gas plants, steamship, and miscellaneous trade. Although the production is smaller in division 2, more men are employed, the ratio being roughly 3 to 2. To show the methods of mining that depart somewhat from the ordinary methods, mines have been selected that, because of physical conditions, present problems that are typical of the district.

Mining Methods in Pierce County

Nature of Coal Seams Worked

The method of working steep seams described is practiced, with slight modifications, in the different beds of the Pierce County coal measures, especially where explosive gas is likely to be met. In general, the method is always employed where gas is found and where the roof and bottom are troublesome. It is used on all dips where the coal will run, that is, on dips to 65°; whenever a dip exceeds 65° to 70°, the working place becomes unsafe and the bed is hard to work. In such cases, chutes are driven across the pitch to give 45° to 50° pitch. The coal is mined by the chute-and-pillar system, for the character of the walls is such that mining by wide breasts is usually impracticable. The coals do not fire spontaneously. The seams worked by this method vary in thickness from 3 to 25 ft., although in Pierce County they do not reach the maximum thickness.

Method of Development

The gangway, Fig. 2, which is the intake airway, and the counter gangway, which is the return airway, are driven parallel. The counter gangway is up the pitch from the gangway and the stumps are from 20 to 50 ft. thick. Chutes are driven 6 ft. wide from the gangway to the counter on 50-ft. centers, from which point they are driven 8 to 10 ft. wide up the pitch to the top of the block. Lifts are arranged so that the chutes will be about 400 ft. long. It has been found impracticable to make blocks over 50 ft. in length, although the law permits a distance of 60 ft. Crosscuts are driven, varying from a hole just large enough to crawl through to 6 ft. wide. Between every other chute, a "half chute" 6 ft. wide is driven from the main gangway to the counter gangway. Where much gas is encountered or conditions warrant such action, half chutes are driven between all the chutes. Each half chute is a traveling way between the gangway and the counter gangway, and from it access can be made to two chutes at all times.

As shown in Figs. 2 and 3, a board brattice is carried from the counter gangway to the face of the chute. The inby compartment is used for a manway. The boards are nailed on the coal side to a line of props up the center of the chute; these props are set from 2 to 4 ft. apart, depending on the character of the walls. Hand rails are fastened to the props

on the manway side and steps on the bottom are made by placing a prop on the upper side of the brattice props, setting one end in a hitch in the rib. Often a ladder is laid on the floor of the manway and nailed to props, which are placed on the floor at regular intervals to be used for steps.

Ventilation

The details of ventilation are shown in Fig. 2. The main ventilating fans are usually of the exhaust type and the main haulage roads are used as the intake airways. In all mines where explosive gas is generated, approved electric lamps are used by the workmen.

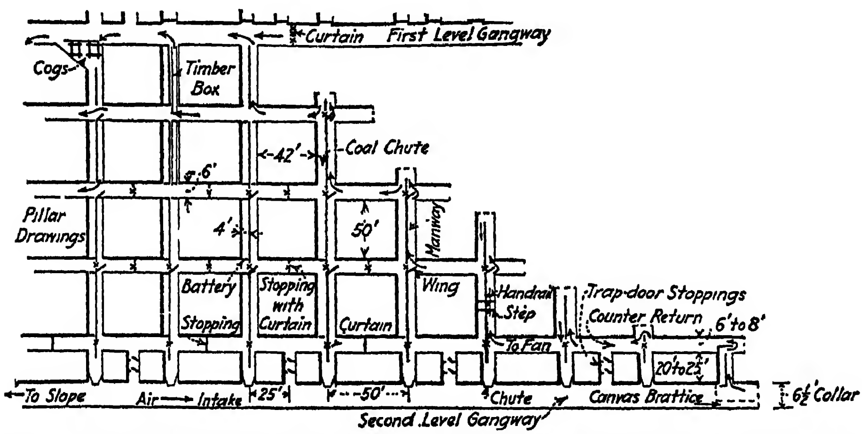

FIG. 2.—DEVELOPMENT BY CHUTE-AND-PILLAR SYSTEM AND DETAILS OF VENTILATION PRACTICED IN PIERCE COUNTY DISTRICT.

The chute between the gangway and the counter gangway is always kept full of coal, thus preventing leakage of air from this source. At every crosscut and on the counter gangway, canvas curtains 2 by 2 ft. are hung in the brattice: access across the chutes can be made through these openings at all times. At each crosscut, wings are built of boards from the bottom to the roof on the manway side of the chute, starting at the chute brattice below the canvas curtain and extending to a wing post set in the center and slightly back in the crosscut. These wings not only deflect the air into the crosscut, but prevent any coal from passing below the crosscut as it runs into the coal side through the canvas curtain, and eliminates danger to men coming up the chute.

Batteries or bulkheads are built at every other crosscut (if necessary, at every crosscut) and the chutes are kept filled with coal. This not only arrests the coal and reduces breakage, but aids ventilation and makes it safe to pass from one chute to another through the crosscuts. A chute starter runs coal from the batteries when necessary.

Blasting

Much of the shooting is done on the solid, which requires that the opening shot be fired first and other requirements of the law complied with. Holes are drilled with ordinary hand machines, which are either post machines or breast augers, although drills of the auger type, such as the Waugh "Ninety," are finding favor where compressed air is available. Power drills are a distinct advantage, especially in a thick seam or a high chute where it is difficult to set up a post machine and the coal is too hard for a breast auger. In some instances, half of the miner's time is used in drilling and this time can be reduced considerably. Compressed air in the place also aids ventilation. In most cases only one miner works in each chute. In one seam averaging 4 ft. and a coal of average hardness, the rate of advance in 8-ft. chutes is 8 ft. per shift of 8 hours.

Drawing the Pillars

In Fig. 3 is shown the method of pillar drawing. Four chutes are usually driven to the level above in advance of the pillars, three of which are usually working at one time. The pillars are drawn on the inby side of the pillar as follows:

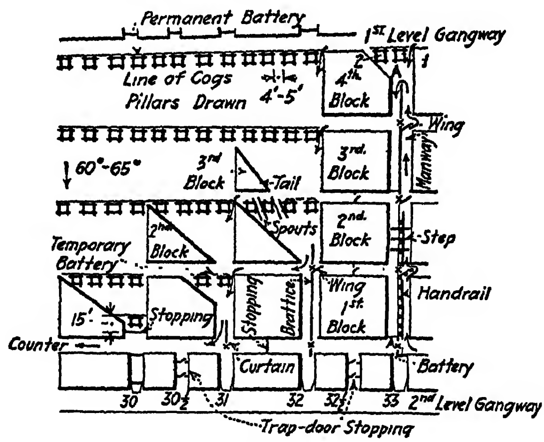

FIG. 3.—PILLAR DRAWING IN CHUTE-AND-PILLAR SYSTEM, PIERCE COUNTY PRACTICE.

Pillar No. 30 is finished. When a pillar is to be drawn, the crosscuts of the chute are well timbered and a cog, 1 in pillar 33, is built on the under side of the upper or crosscut as the case may be; often right up against the face or top of the workings. This cog is placed as close as possible to the inside rib of the chute. The corner A of the fourth block is then worked off and, as soon as space permits, a second cog 2 is placed 4 to 5 ft. from the first. A temporary battery of props is built above this cog to aid travel across the face and to prevent anything falling from above and injuring the men below or knock-

ing out the props. The cogs and batteries are built of 6- or 7-ft. props, depending on the thickness of the seam.

Attacking a block in this manner is called "taking off the angles." The process is continued until half of the block is worked off, as in block 2 of pillar 31, leaving what is called the "tail." The same course is pursued in the next block below, as in 1 of pillar 31, after which the tail or half of the block above is worked and run out. When a tail is to be run out, as shown in block 3 of pillar 32, part of which has been run out, the temporary battery above the cogs is replaced by spouts or wooden chutes. The coal is then run out between the cogs through the spouts, after which a permanent battery is placed above the cogs. In this way the pillars are drawn to the first block, from which only an angle is taken off as shown in pillar 30, and a cog and permanent stopping are placed in the chute neck. By working as shown, the roof breaks above the cog lines and is held up by the cogs. With good roof, it is not unusual to recover 90 per cent. of the coal.

Timbering

The timber is cut on the surface, at the mines; the lagging is split or sawn timber. Split lagging is the best in gangways, heavy ground, and in chute batteries or bulkheads. Ordinarily, the gangway sets are made of heavy poles, set 6 ft. apart, and consist of two legs and a collar, above which split lagging must be placed to safeguard against the coal or shale roof. Ties are usually made of sawn lumber, the size and spacing varying with the gage of the track.

Timber, where possible, is brought down from the surface to a gangway or top counter gangway above the active mine workings. Chutes are driven from the working level to this gangway, or top counter, and one of these is used as a timberway from which props are taken to all parts of the pitch.

On the pitch, props are set 3 ft. or more apart, depending on the walls, above which cap pieces are placed if the roof is bad, otherwise they are set in a hitch. They are slightly underset on the top or bottom, depending on which wall is apt to move ahead of the other; if the bottom is bad, sills are used.

Transportation

Where the mines are opened as slopes, the hoisting is done by means of steam or electric hoists. In small mines, unbalanced or single-rope hoisting is used; but in the large operations, the partly balanced or two-rope hoisting system is common. The loads hauled vary according to the dip and size of the mine.

Mine cars of from 1 to 2 ton capacity are used and, except for some short hauls when mules are used, haulage on the gangways is performed with electric locomotives of both the trolley and storage-battery types.

As the cars are loaded from the chutes on the gangway and the coal spills more or less over the sides of the car, it has been found economical to place the drainage ditch on the hanging-wall side of the gangway.

Crosscuts on Extra Heavy Dips

When the dip exceeds 65°, it is advisable in most instances to drive angle chutes where the chute-and-pillar system is used. The question arises whether the crosscuts should be driven at right angles to the chutes, that is, angle crosscuts, or driven on the strike of the seam. The general practice favors the level crosscut, regardless of whether the mine is gaseous or not. Probably the determining factor in working a steeply dipping seam is the distribution of timber and material on the pitch, and this is greatly facilitated when the crosscuts are driven on the strike of the seam. Other advantages of the level crosscut are: better escapeways for men in pillar drawing, better opportunity to keep the places clear of gas, and greater ease in traveling. A disadvantage is the shoveling necessary when driving the crosscuts. The advantages of the angle crosscuts are that no coal must be shoveled and if desired they can be used as chutes at any time, especially when drawing the pillars. However, the crosscuts must be well covered so that no coal runs into them. There are times when it is a distinct advantage to drive angle crosscuts even when the chutes or breasts are driven on the full pitch when the dip is less than 65°; this is described under the Newcastle operation.

Unusual Methods Employed by Carbon Hill Coal Co.

The mines of the Carbon Hill Coal Co. are located at Carbonado, Pierce County, Washington, on the Northern Pacific Ry., 53 miles from Seattle. They are typical of the district. The mines, because of folding and faulting and the topographic and physical conditions of the seams and walls, present many interesting and complex features of operations; all angles of dip are found and the methods of working the seams vary accordingly. On this property will be found all seams of major importance that have been developed in the district.

The Carbonado mines have been the largest producers in the district; their total output has been over 7,500,000 short tons. Normally, about 1000 tons per day are produced. The product of the mines is a high-grade bituminous coal ranking with the best in the northwest.

Method of Entry

All the Carbonado mines have been opened as water levels starting above high-water level in the Carbon River canyon. This canyon, in places, is 50 to 80 ft. wide and is about 400 ft. deep near the openings. The river has cut through the sedimentary rocks and has exposed the seams.

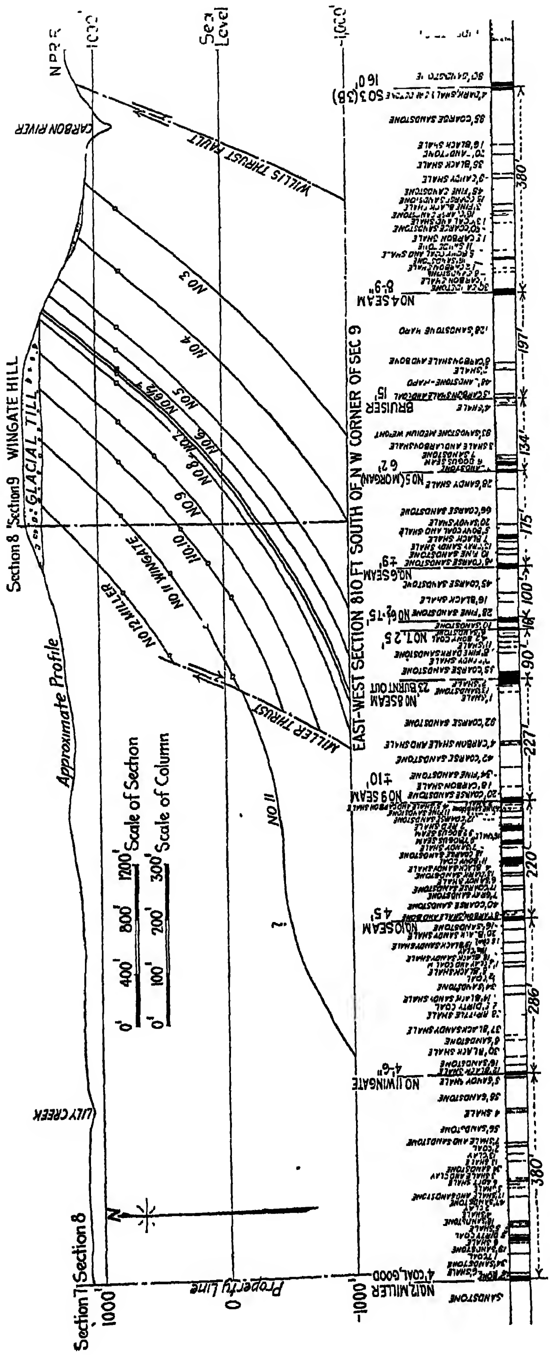

Fig. 4.—Cross-section and column of measures in South mines, Carbon Hill Coal Co.

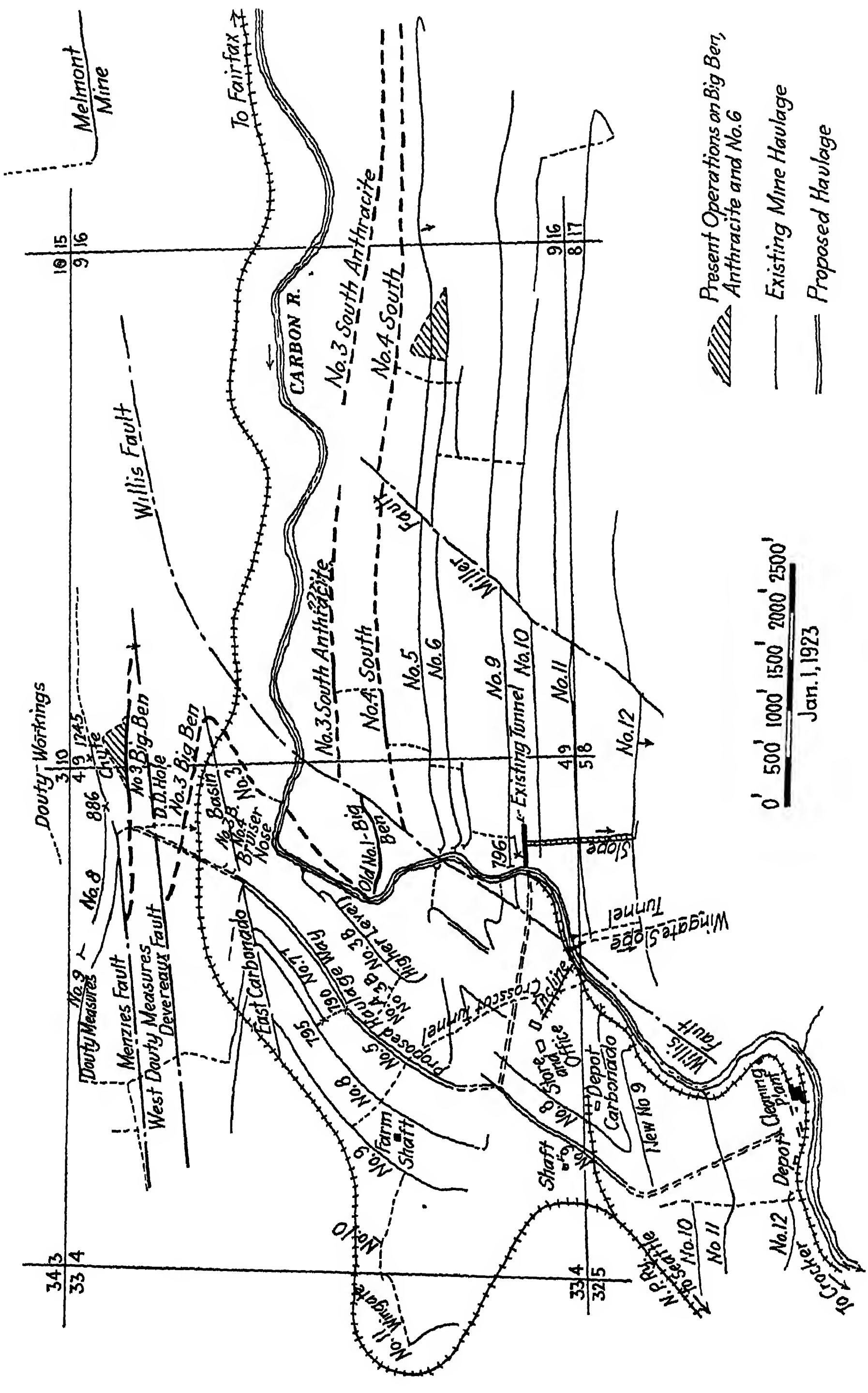

FIG. 5.—WATER-LEVEL WORKINGS OF CARBON HILL COAL CO.

Twelve workable coal seams appear in this series, first identified by Doctor Willis as a continuation of the series at Wilkeson, Burnett, and Spiketon. The tops of the seams have been eroded and the outcrops are usually covered with gravel. Because of the available coal above water level, only two seams, No. 12 (Miller) and No. 11 (Wingate), have been worked to any extent below water level. All the Carbonado seams that have been worked are shown in Fig. 5.

Structural Geology

The structure of the Carbon Hill Coal Co.'s property is divided by the Willis fault into very distinct parts. North of the fault, the structure

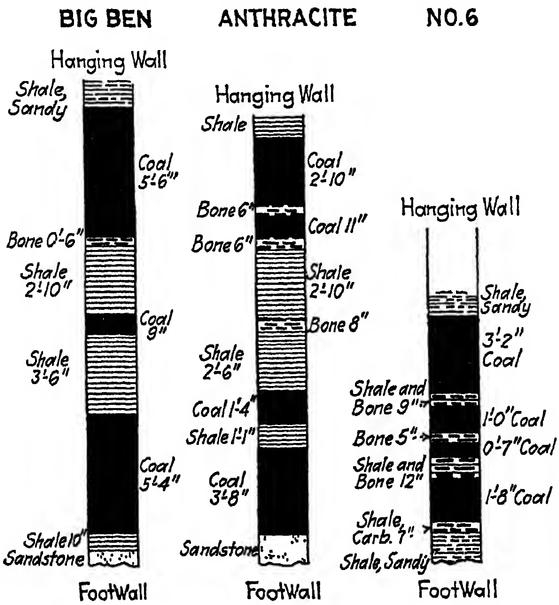

Fig. 6.—Cross-sections of coal seams at Carbonado, Wash.

is one of intense folding and thrust faulting; south of the fault there is only a monoclinal dip to the west. In this section, the whole series appears and is disturbed only by the Miller fault and a small fold in the southwest.

Dikes and sills of igneous rock have been intruded into the measures in sections 9 and 16. Bed No. 8 has been entirely burnt out by small sills and dikes. The presence of anthracite coal in south No. 3 seam, the equivalent of Big Ben, is due to the intrusion of a large sill into the measures below. Fig. 6 gives sections of some of the seams.

Working a Steep Coal Seam by the Longwall Method

The methods of working seams No. 6 and No. 12 (Miller) at Carbonado were changed to a modified longwall method in preference to the

breast-and-pillar and chute-and-pillar methods adopted by the other mines.

The writer had occasion, in 1916, to note the successful operation of the longwall method as applied to a short lift in seam No. 6, and to seam No. 12 during a period of two years, when the level on which the system was being worked was worked out. Several attempts were made to work seam No. 12 by the breast-and-pillar and chute-and-pillar methods, but they failed because the roof could not be kept up. Under J. F. Menzies, a successful longwall method was developed.

The coal of the Miller bed, shown in Fig. 7, is from 4 to 4½ ft. thick and is used as a domestic and steam fuel. The seam is reached by a rock

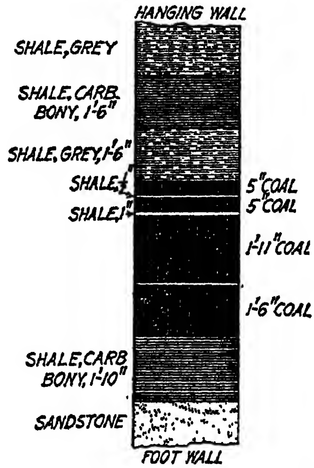

Fig. 7.—Section of Miller bed, Carbonado.

tunnel 600 ft. long from the Wingate slope on the second level. Because of the impossibility of profitably working the seam by breasts or chutes, little has been done on the seam compared to that done on the other beds, and the second level is the lowest level worked. The dip here decreases south from the rock tunnel to the slope, the coal having an average dip of about 38°, which is a little too flat for the best results.

Development

The level worked at the time was originally opened for chutes and pillars, consequently the practice adopted was not the same as will be used in future development. The gangway, which is the intake airway, was driven in the coal and timbered by three-piece sets consisting of 9-ft. legs, 8-ft. collars, and with lagging between sets.

The air travels from the gangway to the longwall face, through a counter gangway, 4 by 4 ft., driven parallel to the main gangway and about 25 ft. up the pitch from it. The air then circulates up the longwall

face to the old gangway above. It was practically impossible to keep this gangway open as a return airway, so a rock tunnel was driven in the foot wall parallel to the top gangway and about 20 ft. from it, as shown in Fig. 9. This tunnel is 7 by 6 ft. and was driven for $19 per yard, which would be about the same rate paid at this time for such a tunnel. Crosscuts, 4 by 4 ft., are driven from the tunnel to the top gangway at intervals no greater than 50 ft. to tap the longwall face; their distance apart depends on the condition of the face.

This tunnel is the return airway for the longwall face, and timber, as required, is brought through it and taken down the longwall face by timber packers. The first cost of driving the top tunnel is slightly higher than driving in the coal, but when the cost of retimbering, general upkeep, and value of a reliable and permanent airway and escapeway are considered, the tunnel is the cheaper. Chutes connecting the gangway and counter are driven up the pitch 25 ft. apart, as shown in Fig. 9.

Opening a Face

The method of opening the longwall face, shown in Fig. 8, was as follows: The first two chutes were driven narrow up the pitch from the gangway with a 12-ft. pillar intervening. The crosscuts above the

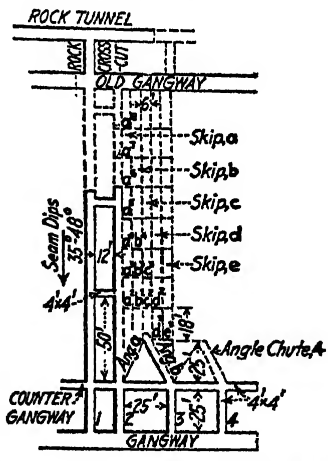

FIG. 8.—METHOD OF OPENING A FACE, MILLER MINE.

counter gangway were driven about 4 by 4 ft. with a 50-ft. block between them. As soon as the chutes were up one block, the longwall face was begun, there being room for one miner, who started in chute 2 and drove angle a to meet angle b, which was simultaneously driven by the counter miner before he proceeded with the counter gangway. Skip, or slice, a, about 6 ft. wide, was then continued up the pitch, its faces (a^2, a^3, a^4, a^5) being at all times about 18 ft. behind the face in chute 2. When skip a was up about 18 ft. from the point of the angle, skip b was started and

continued up the pitch. This sequence was repeated on the skips following. When skip *a* reached point a^5, the face was as shown by the heavy line a^5, b^4, c^3, d^2, e^1. A miner then proceeded to drive an angle from chute 3, to tap angle 4, and the longwall face proceeded.

Before chutes 1 and 2 tapped the gangway above and before the rock crosscut was driven from the rock tunnel at top of chute 1, the air traveled up the last crosscut from the gangway to the counter gangway, back along the counter gangway to the longwall face, up the longwall face and then down chute 1 to the counter gangway, through which it returned to the return airway and to the fan. The usual practice in this field, when the gangways are driven as water levels or have a gangway above to be used as an aircourse, is to drive but one rock tunnel connecting the seams and then drive a chute, or pair of chutes, through to the surface or to the gangway above for a permanent airway. Until this is done, a brattice, flexoid tubing, sollar, or air box is carried in the rock tunnel, making two compartments for the purpose of ventilation. Very often this is connected to the main counter gangway, or airway, but more often a small booster fan is used which is very efficient even for long distances when flexoid tubing is used.

Outside the longwall face in chute 2, no attempt is made to keep this chute open after angle chute 3 is tapped by the longwall face. The chutes 1 and 2 are driven with a small pillar between them in order that they may be rushed, and there is little possibility of profitably recovering the pillar separating them.

Advancing the Longwall Face

The longwall face developed is shown in Fig. 9. The miners took a 6-ft. skip each, keeping about 18 ft. apart and driving through to the top counter gangway or level. As soon as the skip was finished, the miner dropped back to the bottom of the longwall face and started another 6-ft. skip. The longwall face, in Fig. 3, was about 500 ft. long, 30 miners were working upon it, and the output was about 250 short tons per shift of 8 hr. The longwall face, counter gangway and chutes were worked but one shift, while the gangway was operated double shift.

A sheet-iron chute was used to carry the coal from the men to the gangway. This was kept full and batteries were placed at intervals, where necessary to keep the coal from rushing. Four buckers were employed to keep the chute in shape and run the coal. This chute was moved to the longwall face once each week, when the longwall face was not working.

The longwall face advanced inby from 18 to 24 ft. each week, so that the chute ordinarily was never more than 24 ft. from the face. When the chute was moved to the longwall face, a center post was set under each stringer of the sets outby and next to the chute; these strengthened

these sets and protected the chute. The sets outby, and next to the strengthened sets, were knocked out, so that the weight was taken off the faces and the roof was allowed to sag and cave behind, as shown in Fig. 11.

A wing was kept below each miner to prevent things from falling upon the man below, and also to carry the coal into the chute. Owing to the broken ground near the old gangway at the top of the longwall face, cogs were built to keep the top open so that the timber could be brought

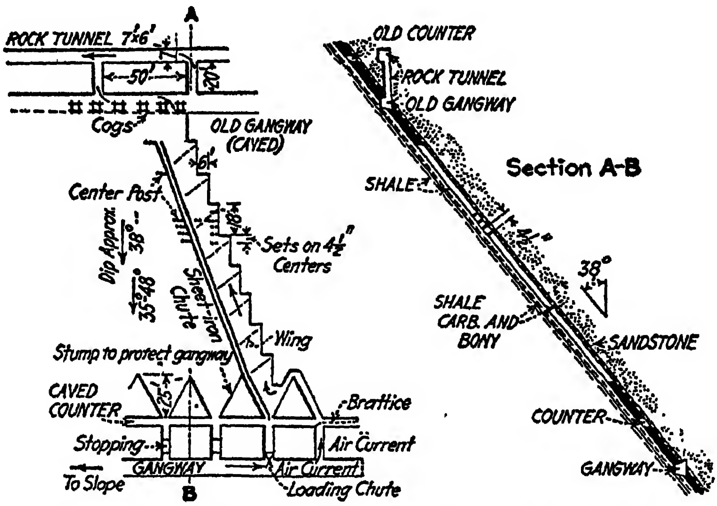

FIG. 9.—PLAN AND SECTION OF MINE WORKINGS, MILLER MINE.

down from above and insure an open place until the next crosscut was open from the rock tunnel.

Timbering

In Fig. 11 are shown the typical timber sets used on the pitch and the lowering of the roof behind the longwall face. These sets are composed of two 4- to 5-ft. legs, usually set in hitches in the foot wall, and a 6-ft. stringer, all of about 8-in. timber. As a rule, lagging is used above the stringers. Each miner, on an average, put in two sets during each 8-hr. shift for which he was paid at the rate of $2.30 per set. The sets were placed on 4½-ft. centers on the pitch and the stringers were set end to end; in the event of a squeeze, a cap piece may be put under the joint and a post set under this, thereby strengthening the joint materially.

As the face advanced before a set, the roof was kept up by lagging and temporary posts. As soon as a sufficient distance was made for a set, a hitch was cut into the rib, about 6 in. in depth, and the end of the stringer was placed in it. A post was then set about 1 ft. from the other end, the other post was set about 6 in. from the rib, and the temporary posts were knocked out. The lagging above the sets can be reenforced if

necessary. The timber on the pitch was taken by timber packers on the opposite shift to that in which the miners work, to each skip face of the longwall by way of the rock tunnel above.

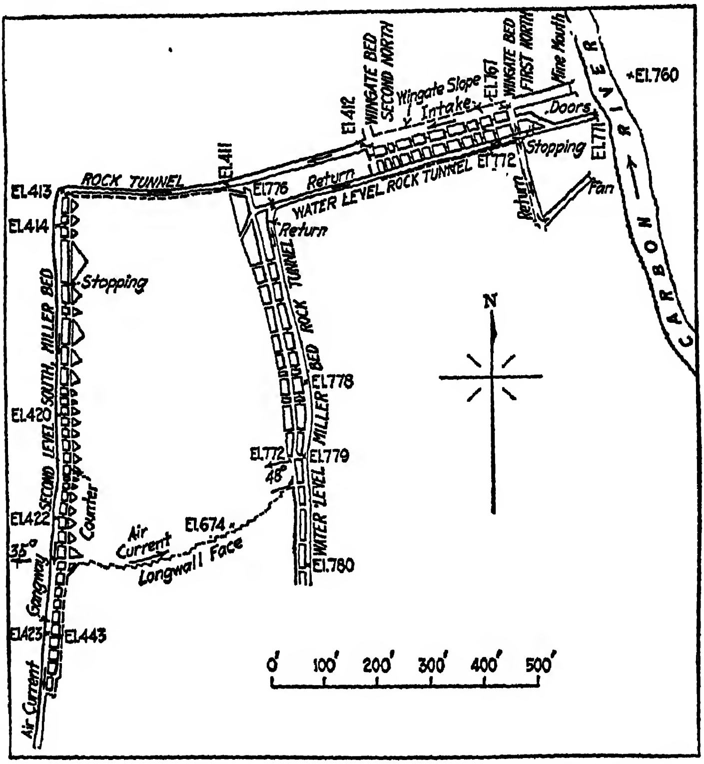

FIG. 10.—MAP OF THE MILLER MINE, CARBONADO, WASH.

Blasting

A good current of air was traveling at the face at all times. Open lights were permitted and the miners blasted the coal whenever they thought it was necessary. As there were always two free faces, only

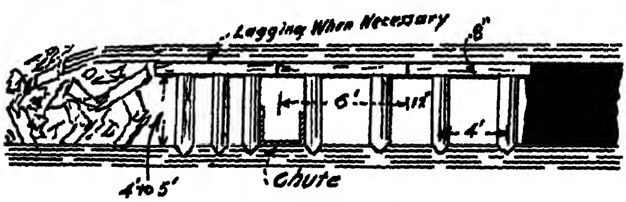

FIG. 11.—METHOD OF TIMBERING AT THE FACE.

small shots were fired. At the time of working this mine, whenever powder was used, it was of the permissible class and was detonated by a No. 7 cap using fuse.

Wages and Hours of Labor

The labor employed at the mines at the time of working this coal seam was all union. The scale of wages was regulated by an agreement between the company and the United Mine Workers of America. In most cases, the miners were working by contract, and the average wages were above the scale rate for day work. The minimum wage was $3.15 per shift of 8 hr. (1916), as compared with $5.25 today (1923). The miners on day work received $3.80 per shift of 8 hr. and were furnished with all tools and blasting supplies, as compared with $6 under the same conditions today, with the exception that the mines in this district are non-union.

Advantages Gained by Longwall Method

It has been proved that, by longwall methods, a larger tonnage per man can be maintained and a larger percentage of lump can be produced. It is stated, however, that the cost per ton is slightly higher than by breast-and-pillar system or chute-and-pillar. This was not true at this mine. Using the other system, the coal was not mined at a profit; but with longwall, the cost was reduced to a point where a profit was made.

This was the result of several causes. What was formerly a safety-lamp mine, because of trouble from gas arising on account of faulty ventilation, became an open-light mine as there were no places for gas to lodge on the longwall face. When breasts were used, the air had to circulate up and down between the crosscuts, which were kept open with difficulty for but a short time; with the longwall method there were no crosscuts, the ventilation was ascensional, and had only one face to sweep. There was no upkeep on the return airway, as it is driven in the foot-wall rock. Although a large amount of timber was required, it was no greater than was formerly necessary, because under the breast-and-pillar system the breasts had to be well timbered until the pillars were drawn. In the longwall method, the timber can be taken to the face more quickly for it must be moved down one face only while in the breasts it had to be distributed through the crosscuts and packed up to the faces. Less powder was used and a larger percentage of lump coal was obtained because there were a greater number of free faces. The work was concentrated and the longwall face permitted a more frequent and closer inspection of the working places by the mine officials. In the longwall system, practically all the coal was recovered, but the breasts could not be kept long enough for even their limit to be reached, and the pillars had to be worked by small skips or lost.

Advantages Gained by Driving Gangways in Foot Wall

When driving gangways in the foot-wall rock, the first cost of the gangway will be higher; but this will be overcome by the increased

recovery of coal above this passage, the gangway will serve as a permanent airway for the level that may be driven below, and will require but a small upkeep, little retimbering being necessary. Ordinarily, no timber is required when driving is done in the foot-wall rock and the tracks will always be in good shape.

Discussion of Longwall Methods in Pitching Seams

If larger coal were the only factor to offset the higher longwall cost under ordinary conditions and the profits were increased as the result of higher sales realization, it would naturally leave nothing to be desired. However, in a seam of this kind, anything that increases tonnage per man per day will lower cost correspondingly. The illustrations in the foregoing description show that there are no pack walls, as in a regular longwall system, no roadways to be maintained to the face, no brushing is necessary. If the method can be pursued, unless dangerous and uncontrollable caves prevent its operation, less timber is required. The face must be kept advancing and open, as developing a longwall face for ventilation and working is slow and expensive. If squeezing is troublesome, due to the roof sagging or the bottom heaving or sliding, or if there is a combination of these characteristics, such a pitching seam can be profitably worked by a longwall method if it can be worked without loss by any other method. In the writer's opinion, if such a seam is worked with a face not exceeding 400 or 500 ft., lower costs will result if the mine is worked steadily enough to keep the face open and the timber can be easily distributed from the top and through a counter or old gangway or similar opening.

The question is raised why the system is not used more and why it has been discarded in several instances where it has been tried. The writer has studied some of the cases and offers the following suggestions: Probably the two major causes for such failures as have been observed are: first, inexperience and lack of interest on the part of the immediate officials in charge; second, the failure to substitute the contract system in place of day work, a difficulty that may be due to the attitude of the labor union. It is necessary to keep the face advancing and to have regular shooting times. Timber is most efficiently distributed on the opposite shift from that on which the miners are employed. Congestion of coal in the delivery chutes is a drawback and can be avoided only by keeping chutes cleared as coal is made. Chute starters are required to look after coal in the chutes, as congestion will surely result if the face is too long. One chute with two outlets will handle 200 to 300 short tons per 8-hr. shift on a longwall face 400 ft. long.

The writer has never seen a longwall face worked successfully on steep dipping seams when the face is carried up the pitch, as in the method of overhead stoping in metal-mining practice. Shales and

sandstones with more or less carbonaceous material separating them do not permit this method, at least in this district. As the coal is withdrawn, the roof breaks; and if it be strong, there is an area open that is entirely too large to be safe. A cave is almost sure to follow and it breaks along the face, which is lost, causing a wild and dangerous place. There is also greater danger in facing large niggerheads that the coal might contain. It appears that the face must proceed inby or outby along the level haulage road, and not be worked in sections up the pitch, which approaches a wide breast. An area once removed should be of no further use and the quicker it can be allowed to fill with waste the better.

Working a Thick Seam of Two Benches on a Heavy Pitch

Location and Description of Seam

On the north and east side of the Carbon River, seam No. 3 (Fig. 10) is known as the Big Ben and is a high-grade, coking, bituminous coal. Operations were started on the seam in the West Douty measures in the southwest quarter of section 4 and extend south into the northeast quarter of section 9.

On this side of the river, an average section of the seam has 5 ft. 6 in. of coal in the top bench, 5 ft. 4 in. in the lower bench and 7 ft. 7 in. of parting, which is mostly shale (see Fig. 6), although it contains some bony coal and carbonaceous shale. This parting must be left in the mine as it constitutes waste that would be prohibitively expensive to transport and remove in the cleaning plant.

The hanging and foot walls are both good; and if it were not for the extreme thickness of the seam, the inability to gob the waste, and the pitch of the seam, the ordinary prop and cap, or single stick, method of timbering and working would fill all requirements so far as the main walls are concerned. As the pitch varies from 45° to 70° and the lift is approximately 900 ft. long, an entirely different method of working than has been practiced in the field heretofore is necessary.

Factors Deciding Method of Working Adopted

To mine each of the benches of coal in this seam separately naturally raises the question as to which, the upper or lower bench, should be taken out first, and to what extent the workings of one bench can be kept in advance of those of the other bench. When solving a problem of this nature, several factors must be considered, such as the pitch of seam, thickness of seam and its benches, thickness of intervening strata or parting, the hardness and tendency of this parting to swell or to slide, caving habits in general of main walls, any peculiar features of coal to be worked, presence and extent of faulting of strata, and, a most important factor, the length of the lift.

In this particular instance, the main roof and bottom are fairly good. The roof stands well and the bottom, under normal conditions, does not swell or slide to any great extent. There is little difference in the behavior of the coal benches, although the bottom bench does not work quite as freely as the top. As in any seam, the top bench requires less timber as it has the better roof, and the bottom bench the better foot wall. There is, therefore, little to determine from the individual characteristics of the coal benches which should be worked first. However, small faults cut the seam at various points, horses appear, and the intervening shale parting varies considerably. This materially affects any method and is further aggravated on account of the scarcity of experienced pitch timbermen to repair chutes on this long lift.

The problem therefore resolved itself into how the intervening bench of impurities would act, whether the chutes could be, as a matter of safety, economically kept open long enough to recover the coal, and how the main roof would act and affect the lower seam workings. Experience of the management over a period of two years has proved that with an ample supply of timber such a seam can be profitably mined under normal market conditions.

Chute-and-pillar Method Used

As a chute-and-pillar method played an important part in the various methods of working, this method is described.

The usual method of opening up the pitch and the general practices are the same as have been described, with the exception of differences in the method of driving chutes, conducting the air current, and arrangement of manways. The method of drawing pillars is exactly the same. The method of chute driving is common practice at the Carbonado and Wilkeson mines.

As shown in Fig. 12, the chutes are driven up the pitch 3½ ft. high and 4 ft. wide with no permanent brattice for ventilation. As a rule, the chute is driven on a bottom or top bench of the seam, depending on the nature of the walls, which may both be of coal or bone. However, but one wall is usually of coal and preferably the hanging wall, for the loose material must run over the bottom.

Timbering

Timbering of the chute will necessarily vary with the ground, but in the Big Ben seam posts with a cap piece are set every 5 ft. on the pitch in the chute which is on the bottom bench. These last only for the time required to drive the chute but a few blocks and are principally used to carry the canvas brattice used in ventilating the chute while driving between the crosscuts. They also enable the miner to travel to and from the working face for a distance of one block. A step is hitched

in the ribs every 5 ft., as shown, and a step made at the posts as shown. These are destroyed later by the loose material running down the chute.

Ventilation

In Fig. 12, the arrows show the direction of the air current used for ventilation when the gangway is used as the intake. The mines in the Carbonado district have been principally opened as water levels, for which reason chutes are driven to the surface at certain intervals to be used as timber and air passages. Whether these are used as an intake and the gangway the return, or just the reverse, depends on whether or not the mine is gaseous. If electric haulage is used on the gangways, the gang-

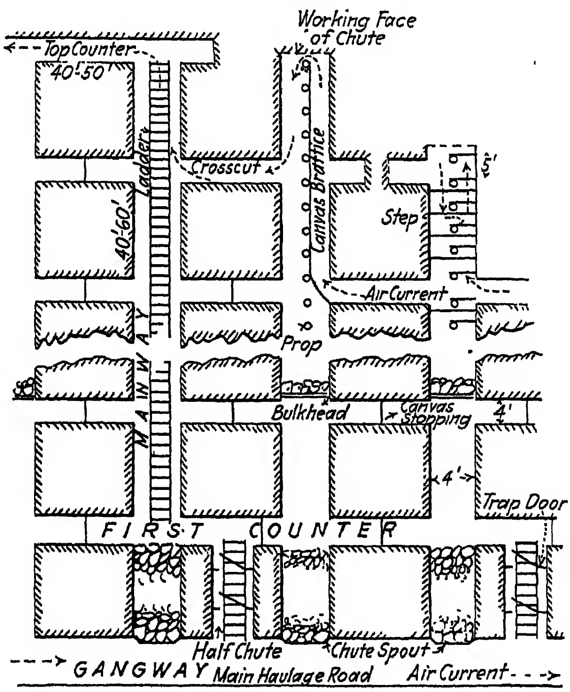

FIG. 12.—METHOD OF DEVELOPMENT AT BIG BEN MINE, CARBONADO.

ways must be the intake airway. No attempt is made to keep the first crosscut or counter gangway open as it is not necessary for an airway other than at the time the area is being worked.

If the gangways are used as intakes, a blower fan and doors are necessary at the main opening (which is not the case at any mine in the Carbonado district), or small exhaust fans are placed on the air chutes of the various seams eliminating the doors on the main haulage road.

This is the general practice here if the gangways are the intakes. The most common practice in the Carbonado district is to use the gangways as the return airways and to place one large exhaust fan near the mouth of the main opening, using doors in this passage and utilizing the air chutes on the different seams as intakes if several seams are worked from one main cross-cut tunnel or opening.

Method of Driving Narrow Chutes

Such a chute can be rapidly and more cheaply driven under the Big Ben method, even if less coal is loosened, provided the chute will stay open and the gas is not troublesome. If the chute will not stay open, it becomes a question of maintenance, and probably a different method of timbering and chute driving would be used. It is impossible to use the chutes for traveling ways, for which reason about every fourth chute is made into a manway and a permanent ladderway placed in it, over which no coal is run during its life as a manway. A timber box, such as is shown in Fig. 2, is placed in the chute, outby, adjacent to the manway chute. Batteries are placed every two blocks to enable persons to cross the chutes safely at these points for any purpose, such as distributing timber.

Pillar coal is the cheapest coal and is usually the source of profit, and the advantage of the system is the speed with which the chutes can be driven. If the chutes stay open, there is a large saving in the timber and timber distribution on the pitch. Because of the nature of the seams, there are cases where a larger chute cannot be kept open and for long lifts it is very advantageous to maintain only a small chute.

Disadvantages of Narrow Chutes

The system has some distinct disadvantages. The work must be well balanced between chutes and pillars as very little coal is obtained from the narrow chutes; if the seam is at all gaseous the faces cannot be kept clear because of the canvas brattice and other ventilation difficulties on the heavy pitch. There is a great deal of trouble from the blocking of the chutes by timber and large pieces of rock, or niggerheads; such chutes must be freed and chute starting becomes a hazardous occupation.

The author does not recall an instance where a chute starter was caught in the wider chutes where manways are kept separate by a brattice in the chute, but in the narrow chutes workers have been suffocated while removing a block. The wide chute offers one remedy for the prevention of accidents from this cause. It is obvious that with a manway in the chute it is easy and safer to take off a board or jar the chute to start the coal, whereas it is necessary to go up a narrow chute, place a

charge of powder, and blast away the obstruction—an extremely dangerous practice if the mine is dusty.

Precaution Used in Chute Starting

The following safe and practical precautions should be taken when starting chutes; in this district, no starter has lost his life when these precautions were taken. As a rule, a chute blocks just below a crosscut, which means that it must be faced for at least about a block. A starter should never go alone to start such a chute; he should have a companion, who remains at the open crosscut below. Before the starter goes up the chute, a grizzly should be constructed over the chute at the lower crosscut, by placing timbers across with openings large enough for the loose fine coal to go through. If the coal rushes or breaks away, as it sometimes does, and catches the starter while in the chute, he goes down ahead or with it and is caught at the grizzly. The fine coal passes through the grizzly and his partner can easily and safely rescue the starter.

Timber Packing

The foregoing method of carrying narrow chutes is practiced in lifts that have reached over 1200 ft. For many years, the workings have been confined to water levels. The timber is usually taken into the mine through combination air chutes and timber chutes driven to the surface. As in other occupations in coal mining, it has been found that when timber packing is done by contract the results are more efficient. The cost of handling timber is an important factor and unless it is carefully supervised the costs are soon on the red side of the ledger.

In the Carbonado district, a counter gangway is defined as a crosscut made large enough to handle timber and material by tramming and which can also be used as a main ventilating passage. These openings are usually made about four crosscuts apart, or about 200 to 240 ft. apart on the pitch.

In the Big Ben mine at Carbonado, the timber packers are not working on contract but are paid a day's wage. Under these conditions, on a pitch of about 60° and where the timber is brought in at the top, or 13th crosscut, which is called the 13th counter, five men can pass 80 props per hour from the 13th to the 8th crosscut or counter. These props average 6 ft. in length but run from 5 to 9 ft. in length. During the 8-hr. day, five men will pass 320 props, 160 from the 13th to the 8th counter, and 160 from the 13th to the 4th counter. It takes nine men 1½ hr. to pass 40 props four blocks through the crosscuts and land them one-half block above or below the crosscut along which they are being passed. In 1 hr., using a timber truck on a counter, nine men can move 40 props along the counter for four blocks and then down the pitch one and one-half blocks.

Methods of Working Tried in Big Ben Seam

Rock Chutes Driven between Coal Benches

Various methods of working the Big Ben seam have been tried, the first of which was to open up the bottom bench as has been described. At first, fourth, and eighth counters, rock chutes were driven to the

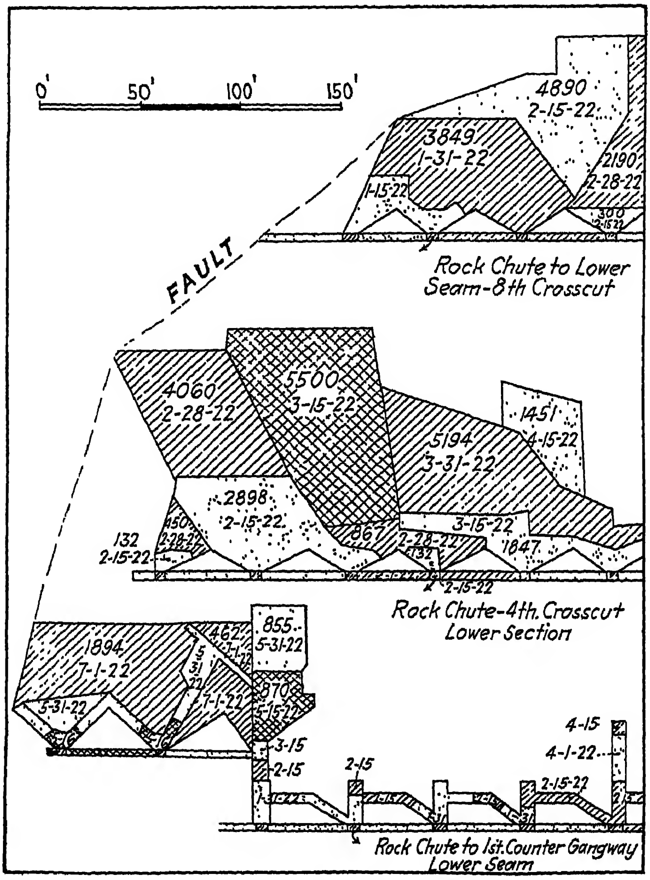

Fig. 13.—Plan of longwall workings, Carbonado mine, on plane of Big Ben seam (top bench); date and progress of yardage areas shown.

top bench, as indicated in Fig. 13, which shows actual pillar measurements in this section plotted on the plane of the seam.

A Modified Longwall Method Used in Top Bench

After the rock chutes were driven to the top bench, the attempt was made to remove the top bench ahead of the bottom bench, using a longwall method on the top bench. The face advanced up the pitch, and failed for the reasons given under the discussion of longwall methods,

but the worst result was the loss of the lower seam. The causes of this were due to the large area worked ahead on the top bench. The coal running down the chutes in the lower bench wore the chute to a width of 15 ft., or more; and because of the heavy pitch, the coal ran against the roof and ultimately wore through to the top seam. The whole area then became wild and uncontrollable, poor pillars resulted and a squeeze started, which overran these workings and extended to the gangway, resulting in a loss of considerable coal and causing heavy maintenance. Much of the difficulty on the lower bench probably could have been avoided in so far as the widening of the chutes was concerned if a large maintenance force of experienced timbermen had been available.

It will be noticed (see Fig. 3) that the top bench was worked in advance of the lower bench for the greater part of the area of the workings, thus leaving large sections of the main roof to act as it would above the parting between the two benches. Here lies the worst danger of this method of working the top bench first, because the main roof does not break immediately and there is no way of telling when it will cave. When the main roof does cave, it breaks through the bench above the bottom-seam workings, caving them in. This happened on two occasions and it was fortunate that no one was in the workings at the time. The method was unsafe and so was abandoned.

Angle-chute Method in Top Bench Worked in Advance

To further test the method of working the top bench ahead of the lower bench, a system of angle chutes and crosscuts was tried on the upper bench. The bottom seam was opened as before and at each block rock chutes were driven through the parting to the top bench or seam.

It was found that whenever a considerable portion of a pillar on the top bench was extracted in advance of a similar operation on the lower bench, the tendency of the foot wall of the top bench or the parting was to bulge and crumble, and thus be liable to slide. In this condition, it afforded poor material for roof protection while the lower bench was being extracted. For this reason, it was found advisable to keep the lower bench workings about one block in advance of similar pillar workings on the top bench.

The chutes and crosscuts on the top bench were driven at an angle of about 45° across the pitch and starting at the top of the chutes of the lower seam, the top seam pillar workings were worked in advance of similar workings on the lower bench for a distance of from one to four blocks. The same results were obtained and a cave from the main roof broke through the parting between the seams and nearly resulted seriously. Experience has demonstrated that this practice is unsafe as there

is no way of telling the condition of the main roof once the top seam is removed.

It was found, over a period of one year, that when the top seam workings were worked ahead of the lower seam workings by the longwall and angle-chute methods described, the recovery was 44 per cent.

Bottom Bench Worked in Advance and Angle Chutes in Top Bench

Because of the experience just described, it was decided to work the lower bench in advance of similar workings on the top bench by starting at the top crosscut and working the lower bench pillars one block ahead of the top bench pillars and, further, to open up the top-seam chutes only as required and drive these chutes on an angle across the pitch. Accordingly the chutes were opened on the lower bench and rock chutes driven to the top seam as required, starting at the top of the pitch workings. The pillars on both benches are recovered by the angle-and-tail method, or regular pitching seam practice, which has been already described.

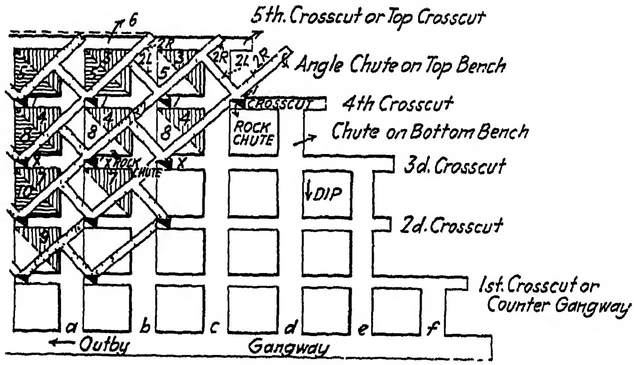

FIG. 14.—DETAILS OF PILLAR DRAWING WITH ANGLE CHUTES, BIG BEN SEAM, CARBONADO.

The mining method is shown in Fig. 14. A rock chute 1 is driven from the next to the top crosscut of the bottom-bench workings between chutes *c* and *d*, and slightly to one side of chute *c*, so as not to weaken the chute and to facilitate traveling to and from the top-bench workings. These rock chutes serve for ventilation and passages through which the coal from the top-bench workings can reach the lower bench chutes through which the coal is delivered to the gangway.

Angle crosscut 2*L*, on the top bench, is then driven with about 45° pitch toward 2*R*, previously driven from rock chute 1 of chute *b*; this makes a connection for ventilation. Angle chute 2*R* on the top bench is then driven. The half-block 3 on the bottom bench is then removed; this is called taking off the angle. A similar angle 4 is then removed on the bottom bench. The half-block 5 between chutes *a* and *b* is then

removed; this portion of a pillar is called the "tail." Block 6 on the top bench is then removed through chute 2*R* from rock chute 1 in chute *a* of the bottom bench. As soon as the tail 5 between chutes *b* and *c* is removed, the V-shaped piece of coal on the top bench vertically above this tail, and between angle chute 2*R* and angle crosscut 2*L* of the top bench will be removed. As soon as the tail 8 of chute *a* on the bottom bench is removed, the coal of the top bench vertically above angle 4 will be removed from angle chute 2*R*2 and rock chute *x* in chute *a* of the lower bench. The workings are advanced inby in the same manner, the pillar workings on each bench retreating toward the gangway.

By using the angle system, it was expected that slides would be averted in the top-bench chutes. Although the bottom heaved somewhat in the top-seam chutes, this was not serious in a distance of one block; but when driving the angle chutes, the high ribs sloughed off and the chute ribs, although lagged, would run on the high side. The high rib sloughs in a level crosscut and this coal can be allowed to stay there, but this is not so in an angle crosscut or chute on this pitch. An excessive amount of extra timbering is required.

Angle Chutes and Angle Crosscuts Abandoned

The method of driving angle chutes and crosscuts on the top bench was abandoned and the present method of driving the top-seam chutes straight up the pitch was adopted. After several months' trial, the method has demonstrated its superiority over the former methods. The most successful method developed is to open the seam by narrow chutes and work the pillars by the usual method, but to work the bottom-bench pillars first, one block in advance of similar workings on the top seam, starting at the top and running the coal down through the chutes on the lower seam. The top-seam chutes are offset about 6 ft. from the lower seam chutes, are driven straight up the pitch, and are opened only as required to keep up with the lower seam pillar workings.

Precautions Taken in Drawing Pillars

Before a pillar is started on the top bench, after the top block in each bench has been removed, the battery holding back the caved material in the lower bench workings is blasted out and the caved material is run into the lower bench area excavated under the area of the top bench to be worked. This caved material is caught by a battery in the lower bench workings above the area being worked here and which is one block in advance of the top-bench pillar workings. This gives a better foot-wall support for the top-bench workings and has been found indispensable for this purpose after one block has been removed and a cave has occurred in the lower level bench workings. However, the workings generally stand open until one or two blocks are removed on the bottom bench.

Caving is then prevented with difficulty and the pillar workings must proceed rapidly to avoid losing a pillar before all the coal is removed, and it is only by filling with the caved material that the intervening rock bench can be kept in place long enough to remove the top bench of coal.

The method of filling the lower bench workings with caved material and drawing the pillars is shown in Fig. 15. In (*a*), the development work for the chutes and crosscuts is complete on the bottom bench, in which *a* is the face of the bottom-bench chute. In (*b*), the top block of the bottom bench has been removed and a rock chute and coal of the top bench has been removed; the same procedure has taken place in the sixth block (*c*). In (*d*), the procedure shown in (*b*) is repeated in the fifth block. The battery at the cog line in the seventh crosscut is then blasted out and the caved material run down against the battery in the sixth crosscut filling the space in the lower bench workings below the coal in the top bench of the sixth block, which is then removed. This procedure

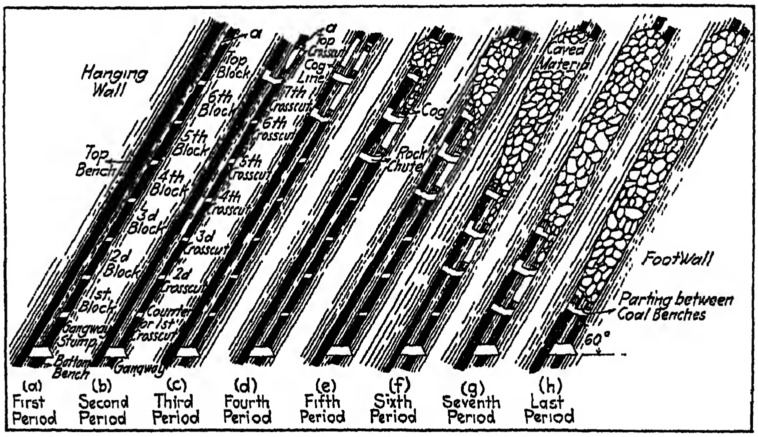

FIG. 15.—TRANSVERSE SECTIONS THROUGH PILLARS OF UPPER AND LOWER BENCHES, BIG BEN SEAM, CARBONADO, SHOWING STAGES OF DRAWING THE PILLARS.

is repeated until the entire section of pillars on both benches is removed, as shown in (*h*).

Over a period of 9 months, it has been found that by this method of working the bottom bench in advance the recovery so far has been 70 per cent. This will be increased for most of the loss to date has been due to the losses in the angle chutes where the ribs ran away. The greater portion of the coal still in is represented by the present live workings and is being recovered.

Discussion of Methods

It is apparent that the regular rock-chute mining method is not strictly followed at the Big Ben mine, but, as in the regular method, all gangways, airways, and counters are developed first in the lower seam.

The question might arise why the top seam could not be worked out entirely from rock chutes on the gangway on the lower seam and then allowed to cave, and the lower bed then worked in the regular manner by narrow chutes and the pillars drawn. Objections to this method are that the parting between the coal benches heaves and slides before the main roof caves, on account of the character of the parting. The presence of small faults and the heavy pitch destroy what is later to be the hanging wall of the bottom bench. The lift is so long that a squeeze comes on the lower workings and the chutes on the lower seam rapidly become uncontrollable; this means the making of manways and timberways on both seams, while the method last described requires this to be done but once, and that in the lower seam.

It is not an uncommon method to work hard firm seams, such as anthracite, on a pitch and have the operations carried on simultaneously in both seams. However, the nature of this seam is entirely different and it is not a common practice to carry on pillar workings in the bottom bench in advance of similar workings in the top bench, as has been described in the foregoing paragraphs.

Conclusions

It is the author's opinion that when the parting between the two benches of coal is firm and thick enough and one or both of the walls at the top bench are inclined to heave or sag to such a degree as to allow the main roof to break over the waste workings in the top seam or bench, an excellent method is offered of avoiding bumps in the workings of either seam or bench in deep mines, when the walls of the lower bench are firm. If the lift is made such that a longwall method, as described in the Miller mine, is used, the main roof will break but owing to the filling of the top bench workings with waste no serious consequences result. The bottom bench can then be worked and caves obtained as desired. However, if this is not done the top bench is usually lost, the lower bench workings remain open for a considerable area, and when the main roof does let go such a tremendous pressure is instantly thrown on the adjacent workings that a serious bump results, crushing the pillars, breaking the timber, and at times caving in the section of the mine. This is especially true in a region of faulting. If the coal makes much gas, a large quantity of gas is often liberated from the crushed pillars at the same time. If the caving of the roof can be regulated, the bottom will not give any serious trouble from bumps.

In a pitching seam, as in any other, when the lift is over 450 ft. long, trouble is experienced in one way or another. From the start, due to the shortage of inexperienced labor, it has been difficult to maintain the chutes and, even with plenty of such labor, the cost of chute maintenance in a seam of this kind is very high. The lift is entirely too long

and the high maintenance cost is largely due to this cause. In the present operation of the Big Ben seam, the gangway has reached its limit and the workings have not been extensive enough to more than demonstrate by experiment the best method of working this seam to follow in future operations. If the present lift were made into two, the present mining system in a new development would make a more profitable mine. Further, the output would be more flexible. Less territory would have to be kept open for the same output or the same territory opened could be made to yield a larger output and what is now a struggle to yield 200 tons per 8-hr. shift could be made to yield 300 tons in the same time at a lower cost per ton.

Machine Mining at Newcastle Mine, Pacific Coast Coal Co.

Location of Mine

The Newcastle mine is located in King County, Wash., 22 miles by rail from Seattle on the Pacific Coast R. R. At present, the output is

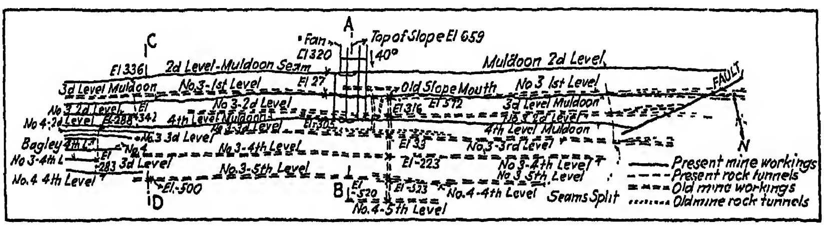

FIG. 16.—MAP OF NEW CASTLE MINE, KING COUNTY, SHOWING SEAMS WORKED.

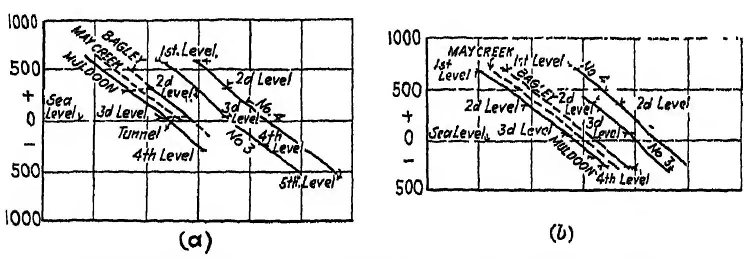

FIG. 17.—(*a*) TRANSVERSE SECTION OF STRATA ON LINE AB; (*b*) TRANSVERSE SECTION OF STRATA ON LINE CD.

about 900 tons of coal per 8-hr. shift, 300 men being employed. Five workable seams are present on the company's property, which is distributed over three sections; viz., 27, 26 and 25, T 24 N, R 5 E, W.M. The seams have an average dip of 40°, and are separated by the usual shales and sandstones. The coal beds worked are shown in Figs. 16 and 17.

Geology

The geology of this area has been fully described.[1] The coal mined is subbituminous and finds its principal market in the Puget Sound cities and contiguous territory.

Method of Development

As the principal operations in the Newcastle mine are confined to the Muldoon seam and, to a lesser extent, the No. 4 seam, and the methods of mining are identical in both, the description of the Muldoon will apply to No. 4. Sections of these two coal beds are shown in Fig. 21.

The character of the walls and the coal of the Muldoon and No. 4 seams are such that breasts varying in width from 15 to 70 ft., can be mined to advantage with a lift from 400 to 800 ft. on the water level.

Experience has demonstrated that this mine cannot, under present conditions, be economically worked mining the coal on the pitch on the advancing method, for the walls, although excellent so long as only the gangways and counter gangways are driven, become very bad and the gangways are squeezed and expensive to maintain if kept open for a long time. The coal in the gangway and counter stumps became crushed to such an extent as to be too fine to be profitably mined; the worked out areas were apt to fire, due to spontaneous combustion; and the amount of black damp evolved and difficulties in keeping the airways open made the ventilation expensive.

Retreating System Used

Because of the foregoing, this mine has been worked on the retreating system. The lateral extent of the mine workings on the 4th level was 8000 ft. from the slope on the east side, and 4500 feet on the west side. The mine has been worked to the 4th level from the present slope, which is about 1760 ft. long and dips about 40° (see Figs. 16 and 17.) A water level was worked and called the first level. It takes about five years to sink the slope one level, a distance of 500 ft, complete the main airways, and drive the gangways and counter gangways to the boundary and prepare the new level for retreating operations. Taking all things into consideration, the gangways are advanced at the rate of about 2000 ft. per year. There is no deviation from the ordinary method of gangway and counter gangway driving already discussed, with the exception that chutes are not opened except at 300-ft. intervals, these being driven to the counter gangway only and used for ventilation and dump chutes for the disposal of the coal from the counter gangway. By this procedure, but few stoppings are required and all of the air possible is conveyed to the inside end of the workings.

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[1] George Watkin Evans: The Coal Fields of King County. Washington Geol. Surv. *Bull.* No. 3.

Ventilation, when developing ahead of the last dump chute, is accomplished by booster fans, electrically driven, and air boxes. The dump chutes are so spaced that they can be later used for regular chutes.

On this dip it has been found that a 70-ft. pillar can be worked, 35 ft. on each side, from 20-ft. breasts. The breasts are therefore opened on 90-ft. centers, except where an old dump chute is utilized, when the pillar is proportioned to suit the conditions. The system of working is shown in Fig. 18.

Method of Driving Breasts and Drawing the Pillars

As shown in Fig. 18, the first breast is driven at or near the face of the gangway, although it is preferable to have the gangway extended far enough beyond the last chute to provide room for several cars to be loaded at the last chute. Breast 2 is also started and the relative positions of

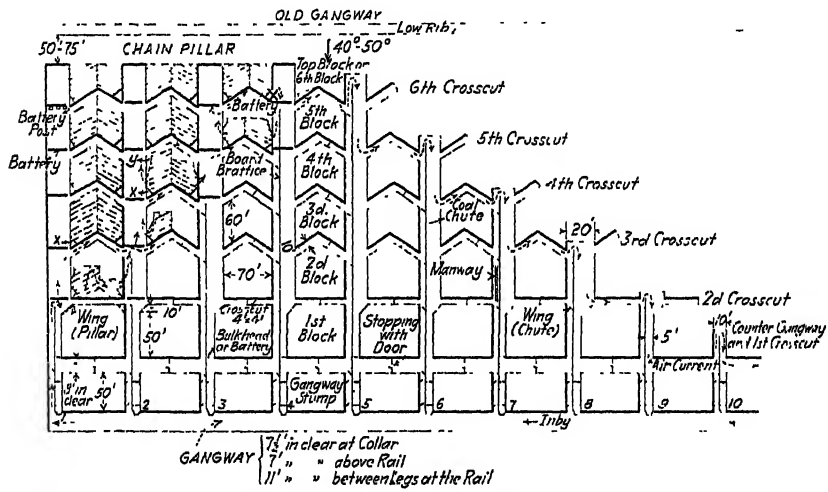

FIG. 18.—DETAILS OF DEVELOPMENT, PILLAR DRAWING, AND VENTILATION, NEW CASTLE MINE.

the breasts advancing up the pitch are shown by breasts 4 to 10, breast 4 having reached its limit, which is from 50 to 75 ft. from the low rib of the old gangway above. The chain pillar is varied in pitch length according to the dip of the seam, tendency for the coal to run, nearness to the face of the old gangway, and amount of water running out of the old gangway that would have to be pumped. The two principal reasons, in their order of importance, for leaving this chain pillar in are to confine as much water as possible on the level above and to serve as a barrier for the blackdamp in the old workings.

A level is always sealed off near the main airways and slope and the blackdamp is confined to the sealed-off areas. Advantage is taken of this condition and pipes, with valves and hose attachment, are placed at

these stoppings so that, in case of a fire, the blackdamp, which in most instances is under greater pressure than that of the mine atmosphere, can be made to flow effectively through pipes to any desired part of the mine, and the workings near the stoppings can readily be flooded with this inert gas.

Details of Breasts

When driving a breast, it is mined 10 ft. wide to a point about 10 ft. from the second crosscut, where it is widened to 20 ft. The manway, with a ladderway, also serves as a passageway for air, timber, compressed air and water pipes. This manway is driven about 5 ft. wide and is separated from the coal side of the breast by a board brattice. The coal side is kept full of loose coal (worked full) up to the top crosscut outby, to prevent breakage and aid ventilation. As no traveling is done in this system, except through the open crosscut at the top and the bottom counter gangway, but one battery or bulkhead, is ordinarily used; this is placed at the counter gangway, as shown in breast 3. However, unless the chute can be kept full, it is advisable to have bulkheads above the counter gangway, at every crosscut, to eliminate the breakage of the coal. It is an easy matter to remove the bulkhead boards holding back the coal if there is enough to keep the entire coal-side full, and to replace them if necessary. The main objection to too many bulkheads is the time lost as the result of large chunks, timber, etc., becoming lodged at the bulkheads when running the coal. Wings placed at each crosscut deflect any loose coal from the manway side through a small opening in the brattice to the coal side of the breast.

Angle Crosscuts Used

Above the second crosscut, and sometimes above the counter gangway, all crosscuts are driven on an angle across the pitch, half way from each side of the block. These are driven as angle chutes 10 ft. wide on a pitch sufficient for the coal to run, generally using sheet-iron. They serve not only the usual functions of a crosscut but as wings or coal chutes when drawing the pillars, for which purpose an ordinary crosscut is worthless. If such an opening, called a wing, is driven at the time a pillar is being removed, the drawing of the pillars is slowed down materially. They are longer and cost more to drive than a level crosscut, but the cost of a wing and a level crosscut more than offsets this additional first cost. The only reason for driving the second crosscut level is to get a hole through for ventilating as rapidly as possible, so as to afford storage room for coal, otherwise not possible.

Drawing the Pillar

As soon as breasts 1 and 2 are finished, the pillar, which is called the top block, is removed as is shown in the top, or sixth, block between

breasts 4 and 5, half of the block being removed from each side, the coal running down the angle crosscut into the breast.

The first thing to do before starting a pillar is to build a battery as is shown in breast 4. On this dip, the batteries are made of heavy posts set from 3 to 5 ft. apart across the breast and a lagging of props is placed above them. Some of the lagging is left off as long as coal is being run from above the battery, which would be the case with the lower battery at the 5th crosscut in breast 4. If a cave is likely to occur, all the lagging can be put in place. Before a battery is left, it is banked with coal on the high side, which serves as a cushion for the caved material to land against. The battery protects the pillarmen from the danger of being shut off by a cave; in addition, the small block *x* is used. This piece of coal is left in, and when the squeeze comes upon it, crushes and runs against the battery serving as a cushion; or it is blasted out for the same purpose before the battery is left. The pillar is then breasted as shown in the third block between breasts 2 and 3, and the top block between breasts 4 and 5, but before it is advanced to any extent, an opening is made to the breast for ventilation.

Whenever possible, the operations are carried on simultaneously from both sides of the block; if not, the procedure shown in the third block between breasts 2 and 3 is followed. A small pillar of coal, *y* in breast 2, is left to hold back any cave that may occur in the breast and as a support for the roof; it may or may not be recovered, depending on the state of affairs existing in the old breast. The pillar face advances as a breast, shown in the top block between breasts 4 and 5 and the fourth block between breasts 2 and 3.

The custom here, as through all the district, is to call a working place a chute, breast, or room as long as the face is advancing, is to be advanced, or is standing idle with the pillars still in. As soon as the removing of a pillar is started the working place is called a pillar; and breasts 1 to 4 would be known as pillar 1, pillar 2, etc. The pillar drawings retreat toward the gangway and the workings proceed outby.

The gangway stumps are removed as soon as block 1 is removed. The method used is to breast up the entire stump. This is done by carrying a face up the pitch from the gangway to the counter gangway the entire width of the stump, or 70 ft., the coal being loaded into mine cars through chute spouts placed apart a distance equal to that between the centers of two cars placed bumper to bumper. In this way storage for coal is provided in the stump area, several cars can be quickly loaded, and the stump removed with the maximum speed.

In Fig. 19 is shown a section of the mine and indicates the progress made in an area selected for the reason that it shows a condition contrary to that desired. In this section, the mine was worked inby and not on the retreat, because there was an unusual demand for coal and the

gangway had not reached its limit. In the workings shown, the mine was closed down for four months, during which interval the pitch workings caved tight although timbered with four-piece sets of 8 by 8-in. timbers, the collar being 10 ft. long and supported by three posts, placed about 6 ft. apart on the pitch, lagged between, and collars end to end across the breasts. The counter gangway was reopened and kept open with great difficulty, and the gangway has had to be retimbered several times. These conditions on the gangway prove that this seam of coal

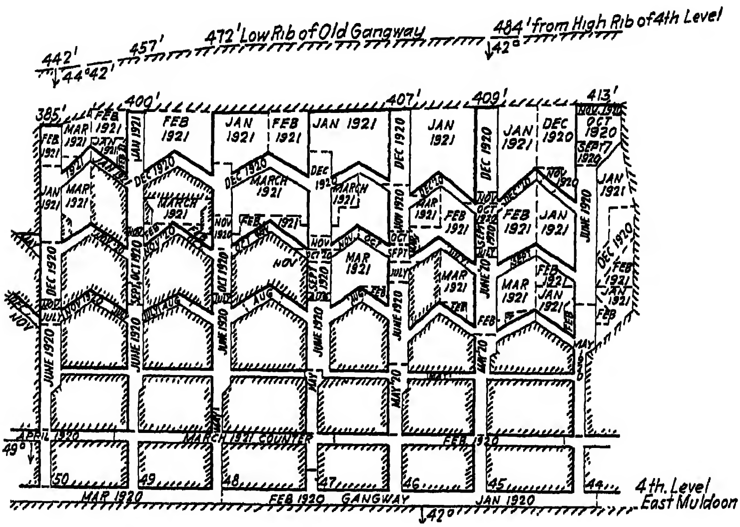

Fig. 19.—Progress map on plane of seam, showing conditions of pitch workings, New Castle mine.

cannot be worked advancing along a gangway of normal length even if large pillars are left in and only the breasts worked. The pillars left in become so badly crushed that the coal could not be profitably mined.

Timbering

Unless the roof is exceptionally good and likely to remain so, the gangways are timbered with three-piece sets spaced 7½ ft. between notches at the collar, 7 ft. in the clear above the rail to the underside of the collar, and 11 ft. in the clear between the legs at the rail.

Where the roof is good, single-stick timbering is used on the pitch. The general practice is to use sawed timber of not less than 7 by 7 in. in cross-section, and cap pieces with the props. If the roof is bad, four-piece sets are used, as has already been described.

All timber used on the pitch is carried by the miners. There are distinct advantages in having this work done by the miners when the latter are working on the contract system, which is the case at this mine. Then each working place gets its proper supply of timber as needed; there

are no timber packers; there are no delays on the pitch due to the passing of timbers; there is no obstruction of crosscuts due to timber being stored in them; and a more accurate record of the timber used and its distribution is possible.

All timber is placed on the gangway in front of each breast on the shift when no coal is being transported and the miners remove most of it above the chute trapdoors before going up the pitch. An order for timber needed in each working place is given by the miners to the district fireboss, who in turn delivers it to the timber distribution supervisor who sees that the order is filled.

To be successful, this system is carried as part of the contract with the miner, who is paid for packing the timber and putting it in place, but receives no payment for timber not in place. Contract rates for timbering and timber packing are discussed under wages and hours of labor.

Mining the Coal

All mining is done on the contract system. Both mining machines and coal picks are used. As the coal works freely when the roof pressure is brought into play, it is not necessary to undercut or shear the coal in the pillar workings with machines, except when driving the pillar wings and removing the top block, or starting the pillar face in the block. The machines are not used in driving ordinary crosscuts and counter gangways, where the shooting is done on the solid and where there is not room to use a machine to advantage. They are used at times in the gangway and always in the breasts, chutes, and angle crosscuts.

Factors Determining Selection of Mining Machines

Since the introduction of fuel oil and the development of hydroelectric power, markets for steam sizes of coals of the subbituminous rank have been closed, necessitating the marketing of this coal as a domestic fuel. This requires a greater production of the larger sizes. As the mines increase in depth, this becomes more difficult and the cost of production increases with the depth unless something is done to offset the added expense.

With the substitution of high explosives for black powder and with a type of miner not skilled in the use of a pick, production increased through the practice of solid shooting, which does not make large sizes of coal. Something had to be done to increase production and at the same time give more lump coal. These circumstances have necessitated the extensive use of machines for doing the work formerly done by skilled miners, namely mining the coal at the face instead of shooting off the solid, also doing it at a greater rate than is possible for even a skilled pick miner. It is a fact that in production alone, with a very keen market, even for fine sizes, the use of machines has decided the difference between the

black and red side of the cost sheet of this mine, on account of the greater progress made.

Method of Operating Machines

The machine used is the Sullivan "Post Puncher." The method of setting up the machine is essentially the same regardless of where it is to operate, but its position is a matter of much importance. Ingersoll machines of the same type were also used. It is advisable to use breasts 20 ft. wide in the Muldoon seam although 40-ft. breasts have been used on the upper levels and on the present level in No. 4 seam. In a 20-ft. breast, there is plenty of room for two miners and not enough for three

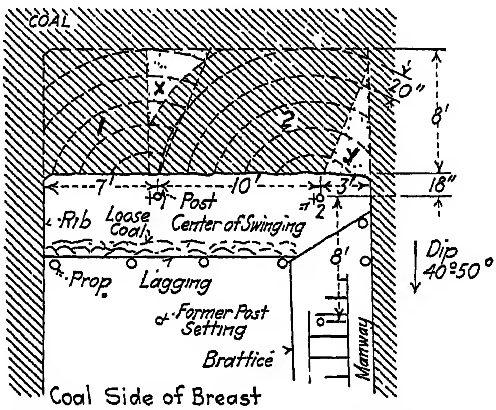

FIG. 20.—MINING WITH THE POST PUNCHER.

to work to the best advantage. They divide their work in such a way that one runs and takes care of the machine, and the other packs and sets the timber and assists the machine miner to move the heavy part of the machine. They work jointly in all operations to their best advantage and divide the earnings of the place equally.

As shown in Fig. 20, the first cut is made from a post set from 18 in. to 2½ ft. from the face and about 7 ft. from the rib of the coal side of the breast, although if the dip of the seam and character of the coal are such that the coal is not apt to break out without warning, the cuts are started alternately at each rib to avoid moving the machine as much as possible. It is good practice to work toward the manway side of the breast and the machine is placed to suit the conditions in the place.

A cut 8 ft. in depth is put in, using different lengths of extension bars. One miner operates the machine, swinging it by a worm crank, with one hand, and feeding the cylinder forward with the other hand. The cuttings fall out of the cut because of the dip of the seam. Time is saved if two posts and sets of blocking are available, and while the machine runner is making the cut the other miner sets up the second post. When the rib cut is completed the transfer of the machine, which

weighs about 225 lb., to the other post is quickly made and the second cut put in.

The machine and posts remain in the breast near the face until the breast is completed, as there is no shooting of coal to injure the machine

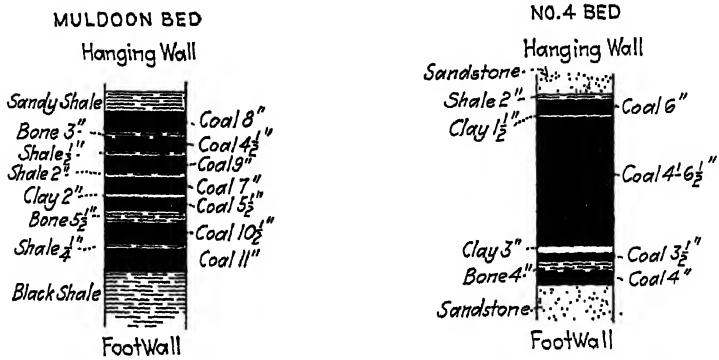

FIG. 21.—CROSS-SECTION OF MULDOON AND NO. 4 COAL SEAMS.

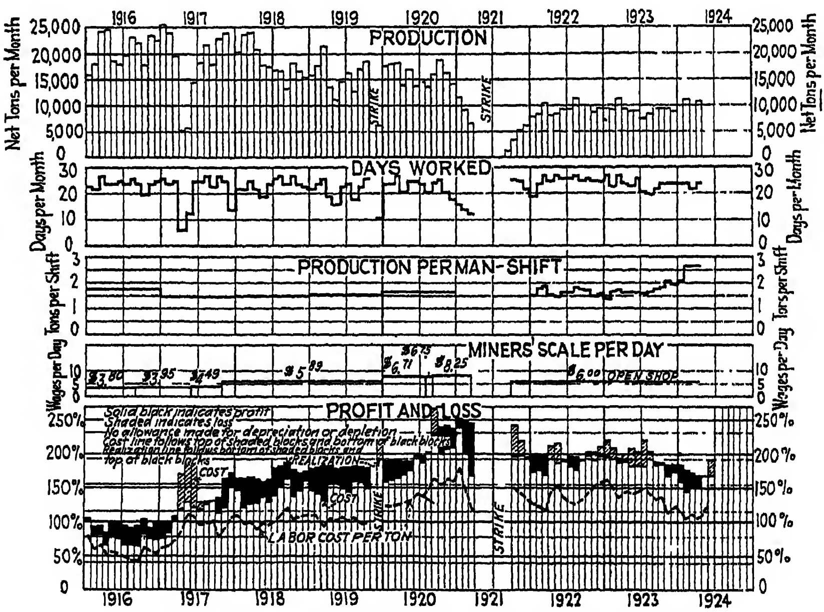

FIG. 22.—OPERATING STATISTICS AT CARBONADO MINE.

nor loading of coal to interfere with it. The main precaution to be observed is the placing of the post so that the coal will not be apt to break out and catch the machine runner. Sufficient coal is kept near the face, by means of lagging above the props, to furnish solid footing for the machine runner. If there is likely to be any danger of the coal

breaking out, the pieces of coal x and y will not be mined; probably they will have to be blasted out.

In mining the Muldoon seam, the cut is made in the band of bone about 5½ in. thick and 2 ft. from the bottom (see Fig. 21). In No. 4 seam, the cut is made in the 6-in. piece of coal near the roof. The position of the swinging gear on the post depends on whether the coal is mined near the top or bottom. Very little blasting is required, for the coal once mined soon works free and can be easily taken down with a pick; if shooting is required, the shots are very light.

Progress Made with Machines

The average number of square feet that can be cut in an 8-hr. shift using mining machines in the Muldoon and No. 4 seams follows:

In a breast 40 ft. wide; 120 sq. ft. for three men, or 40 sq. ft. per man-day.

In a breast 20 ft. wide; 100 sq. ft. for two men, or 50 sq. ft. per man-day.

When using machines, a breast 15 ft. wide is not advisable as it is too wide for a bad roof and too narrow for a good roof.

In chutes and angle crosscuts 10 ft. wide; 60 sq. ft. for two men or 30 sq. ft. per man-day.

These averages include time taken up in the working place for timber packing, setting timber, chute building, and all other work necessary in the place. The results obtained when timber packers were used were much less.

The average number of square feet that can be cut with mining machines in the Muldoon seam when not timbered with sets is:

In a breast 40 ft. wide; 240 sq. ft. for three men, or 80 sq. ft. per man-day.

In a breast 20 ft. wide; 160 sq. ft. for two men, or 80 sq. ft. per man-day.

In a breast 15 ft. wide; 130 sq. ft. for two men, or 65 sq. ft. per man-day.

In a chute 10 ft. wide; 100 sq. ft. for two men, or 50 sq. ft. per man-day.

These averages include time taken up for chute building, timbering, timber packing, and all other necessary work.

DISCUSSION

Eli T. Conner, Scranton, Pa.—In 1910, I made a professional visit to the Carbonado mine, in which, as I understand the paper, most of the work described has been done. At that time the manager was a Mr. Davis, who had spent some years mining in the southern anthracite regions of Pennsylvania, where the coal beds stand on rather steep pitches.

He said that he had introduced in Washington many of the methods generally practiced in the southern anthracite region. While the pitches of the beds at Carbonado were about the same as those of the southern anthracite region, they were thinner than the Great Mammoth bed and some others.

The practices described appear to be accomplishing somewhat better recovery than the average experience in the thicker beds of Pennsylvania. In the steep pitching measures of the Pennsylvania district, it is a general practice to open from the gangway with chutes square up the pitch; then after passing the airway, which usually is 40 ft. above the gangway, and parallel thereto, chambers or rooms are extended to the rise, gradually widening to 24 to 30 ft., which chambers are advanced to the limit, usually from 250 to 300 ft., keeping the chamber filled with coal and carrying a manway on each side. The plans described differ from the Pennsylvania practice, in that chambers to the rise are driven narrow and not square up the pitch, which reduces the pitch of the chutes materially. This practice permits of using an open chute, down which the coal is carried to the gangway. By this practice the yield in large coal is substantially increased, or, putting it another way, the breakage of coal incident to working steep pitching measures with full chambers is substantially reduced by the method described. I have seen this same method successfully conducted at Bankhead, Alberta, where the necessity for care in the handling of coal, by reason of its friability, is imperative.

The practices described are an advance upon the ordinary methods that for many years have been practiced in the anthracite region of Pennsylvania.

Simon H. Ash (author's reply to discussion).—At the present time I do not know of any mine in the heavy dips of western Washington that is working with chutes such a seam that the dip would be from 20 to 25 per cent., as they would be prohibitively long from the standpoint of maintenance. To run cars inclines have been driven on a dip of 20° to 30°. Where the coal is run down a chute, the chute has a dip of 30° to 45°.

In the Pierce County district, the coals are very friable, breaking easily. Although lump coal is desirable, it is not available and the low ash content becomes the desirable factor rather than the larger sizes. The cleaner coal is found in the smaller sizes for where the coal does hold together it is due to a binder of bone or rock. A growing practice is to crush all coal over 2½ in. round opening, to separate the coal from the bone and rock, and then wash the resultant product with fine coal jigs and tables, the latter giving the lowest ash product. This eliminates the expensive rock picking on picking tables, only enough men being used to remove niggerheads and timbers that are in the mine run. A Bradford breaker at one mine is reducing this cost still further. The

following shows the average percentage of sizes as compared with the ash content:

	Mixed Steam	Coarse Steam	Gas Coal
Screen size, inches..................	3/8 to 1	3/8 to 2 1/2	3/8
Percentage yield..................	35.0	4	61.0
Ash content in washed coal.........	13.65	15	12.5

The chutes are worked full more to protect the ribs and brattice of the chutes from rock, niggerheads, and running coal. In some instances, it is necessary to place lagging in the chutes to permit the coal to build up above them so that the running coal will not wear the bottom, causing the bottom to wear and become lost. Further, the chutes 4 to 10 ft. wide are driven at considerable expense and the profit is made in pillar coal. When possible breasts (15 to 40 ft. wide) are worked, as they yield more and cheaper coal; but as a rule the walls will not stand and the maintenance is high.

If the coal is firm enough to yield lump, there is little danger of the ribs sloughing off and angle chutes or breasts can be driven on any grade. However, the system usually followed is to drive a chute or incline on such a grade that cars can be handled; they are then called planes. The rooms are then driven on the strike of the seam and the coal won from these planes and rooms. As in other localities, a set is called a panel or battery. Planes are placed about 600 ft. apart and electric hoists are used for handling the cars on the plane.

Pocahontas Coal Field, and Operating Methods of the United States Coal and Coke Co.

By Edward O'Toole,* Gary, W. Va.

(New York Meeting, February, 1923)

The Pocahontas district occupies the extreme southern end of West Virginia, principally McDowell, Mercer and Wyoming counties, and a part of Tazewell county, in southwestern Virginia.

The first record of the coal field is a report by Prof. J. P. Lesley, of Pennsylvania, about 1880, in which he mentions a coal opening at the upper end of Abbs Valley, Tazewell county, Va. C. R. Boyd, a mining engineer of Wytheville, Va., published a series of articles in 1881, mentioning coal on Indian Creek, Tazewell county, and on Horse Pen creek, which coals, he stated, extended toward the Ohio River. He also mentions the Abbs Valley coal reported by Lesley.

In 1881, F. J. Kimball (who became president of the Norfolk & Western R. R. in 1883), after examining the Scott mine in Abbs Valley, went northward across Abbs Valley Mountain to the place now known as Pocahontas, where he located what has since been called No. 3 vein of Pocahontas coal. In November, 1882, Jed Hotchkiss and others prepared a cross-section of the Flat Top Mountain coal measures, some notes on which were published by Mr. Hotchkiss, between 1880 and 1892, under the title of "The Virginias."

In May, 1881, the Norfolk & Western R. R. was organized through the purchase of the Atlantic, Mississippi & Ohio R. R., an old line running from Norfolk, Va., to Bristol, Tenn., with branches from Petersburg to City Point and from Glade Springs to Saltville. At that time, charters had been granted for three railroads tributary to the New River Valley, but their proposed routes would indicate that the promoters were concerned with minerals, timber, and a connection with the Chesapeake & Ohio R. R., rather than the development of any coal in the Pocahontas district. One of them was projected to extend up Bluestone River a short distance, but not far enough to reach the coal in the Bluestone Valley, and a little grading was done for a narrow-gage road between New River Station and a point in Giles county, Virginia, near the West Virginia line. All charters and railroads in this district were acquired by the Norfolk & Western R. R.

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* General Superintendent, U. S. Coal and Coke Co.

early in 1882, and construction was actively prosecuted on this company's main line, from near Radford, Va., to the Pocahontas coal field during that year.

In the meantime, the Southwest Virginia Improvement Co. began to open coal mines and build coke ovens at Pocahontas and vicinity, so as to be ready to make shipments as soon as the railroad should be completed, which occurred in 1883. Shipments began in June, and during that year 81,800 tons of Pocahontas coal and 23,763 tons of coke were shipped. From 1884 to 1888, the Norfolk & Western extended its lines down Bluestone River to Flipping Creek and up the latter. In 1888 it completed the Elkhorn tunnel and extended its line to Elkhorn, in McDowell County, West Virginia.

About 1886 to 1889, Messrs. McCreath and d'Invilliers made an extensive examination of the Pocahontas and adjacent coal fields for the Norfolk & Western R. R. Co. In 1890, construction began on the Ohio extension, which was completed to a connection with the Scioto Valley & New England R. R. at Ironton, Ohio, in November, 1892. This gave the Norfolk & Western an outlet to Columbus, Ohio, and the West, and also to the eastern sea board at Lambert's Point, Va., where a coal pier had been erected in 1885 for the transfer of coal into ocean vessels. Another outlet was acquired by the purchase of the line from Roanoke, Va., to Hagerstown, Md. The railroad was reorganized as the Norfolk & Western Railway Co. in 1896.

The Pocahontas Coal & Coke Co., incorporated in 1901, is a land holding and leasing company and does not engage in coal mining or in trading in coal or coke. The areas held by this company have been increased from year to year, the present total being 300,000 acres. The largest single lease now is that of the United States Coal and Coke Co. The Crozer Land Association in the early period assembled 16,000 acres of coal land immediately tributary to the Norfolk & Western Ry., all of which is leased to mining operators.

The construction of the Tug River branch of the Norfolk & Western, from Welch, Va., up Tug Fork and tributaries, for developing the leasehold of the United States Coal and Coke Co., was undertaken in January, 1902, and a second section in May, 1903. Fifteen miles were put in operation to and above Gary in June, 1904. Extensions in this valley have been made from time to time, and at present there are 75 miles of railroad on this branch.

Geology of the District

Grabau divides the Appalachian coal fields into two areas of deposition of similar age, as shown in Fig. 1, giving their origin as a former continent, which he calls "Appalachia."

According to I. C. White, State Geologist of West Virginia, "The lowest and oldest rocks are the Mauch Chunk red shales, which crop to the surface in narrow belts along the crest of the Dry Fork anticline in the southeastern portion of McDowell County. The Pottsville measures, or basal formation of the Pennsylvanian, constitute at least 95 per cent. of the outcropping rocks, the entire group being represented in Wyoming and McDowell Counties."

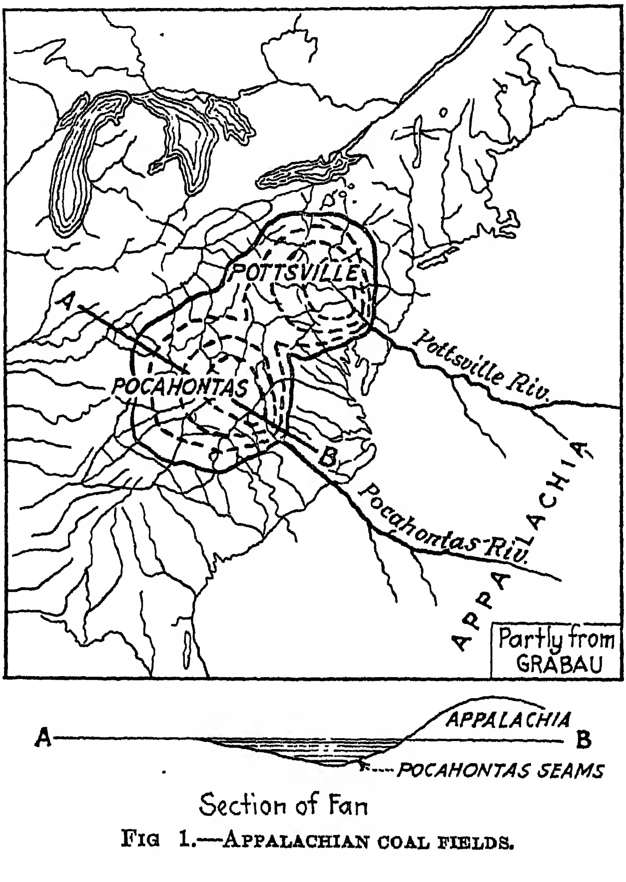

Fig 1.—Appalachian coal fields.

White also gives the sections, names, and elevations of the coal seams of the Pocahontas coal field.

Fig. 2 shows a section (with greatly exaggerated vertical scale) along the Norfolk & Western Ry.; this follows Tug River and Elkhorn Creek, 15 to 20 ft. above their beds, except at the Elkhorn tunnel at Coaldale.

So far as known at present, seven seams in the New River group, namely: Douglas, Iaeger "B," Iaeger, Sewell "B", Sewell, Welch, and Beckley, are of workable thickness in McDowell county, which is considered as synonymous with the Pocahontas coal field.

Only four seams in the Pocahontas group, namely, Pocahontas Nos. 3, 4, 5, and 6, are known to be of workable thickness. No seam of either group is coextensive with the field.

The thickest seam of the Pocahontas series is No. 3, which is 12 ft. thick at Pocahontas, but only 11 ft. in the Elkhorn tunnel, 3 mi. from

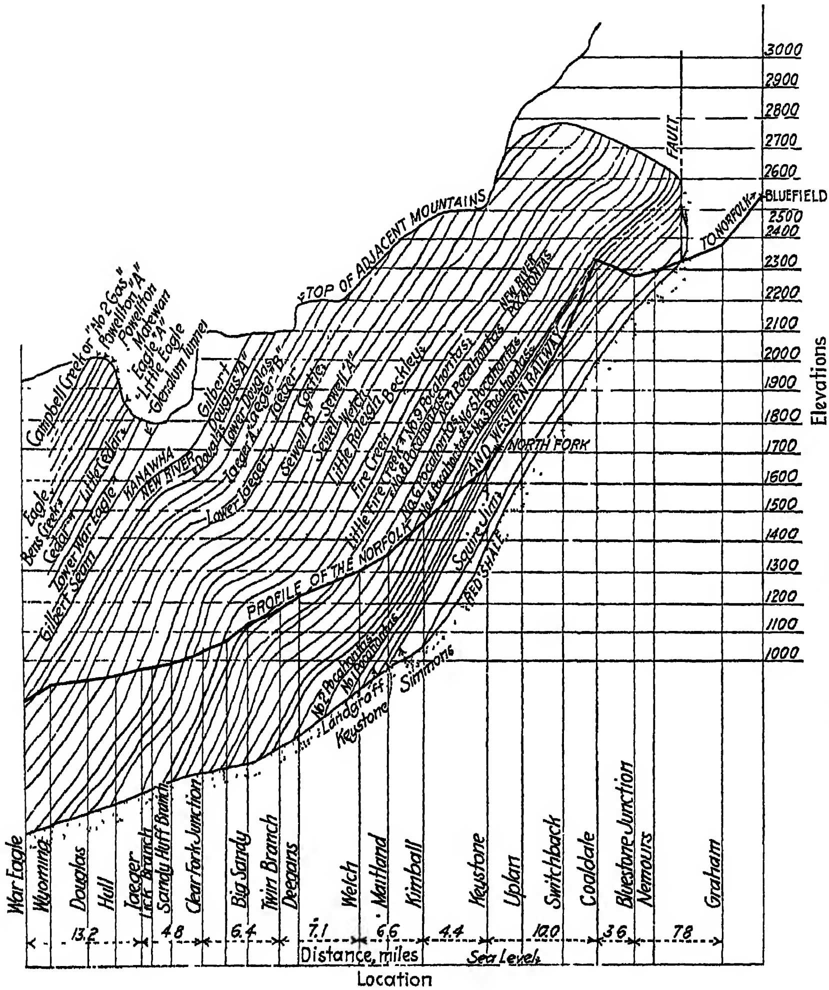

Fig. 2.—Profile of the Norfolk & Western Ry. in the Pocahontas District, in its relation to outcropping coal seams.

Bluestone; it gradually thins in a northwesterly direction until it reaches Kimball, 17 mi. west of Bluestone, where it goes under water, with a thickness of 7 ft. On Tug Fork Branch, on the lease of the U. S. Coal and Coke Co., it runs from 3 to 8 ft. thick.

The No. 4 Pocahontas seam on the lease of the U. S. Coal and Coke Co. runs from 3 to 7.5 ft. thick. These two seams are similar in character.

The workable seams of the Sewell group, immediately above, have a maximum thickness of 5 ft., averaging about 3 ft. where worked.

As will be noted from the foregoing, the Pocahontas coals are found in the oldest coal-bearing measures of the Appalachian coal fields. They

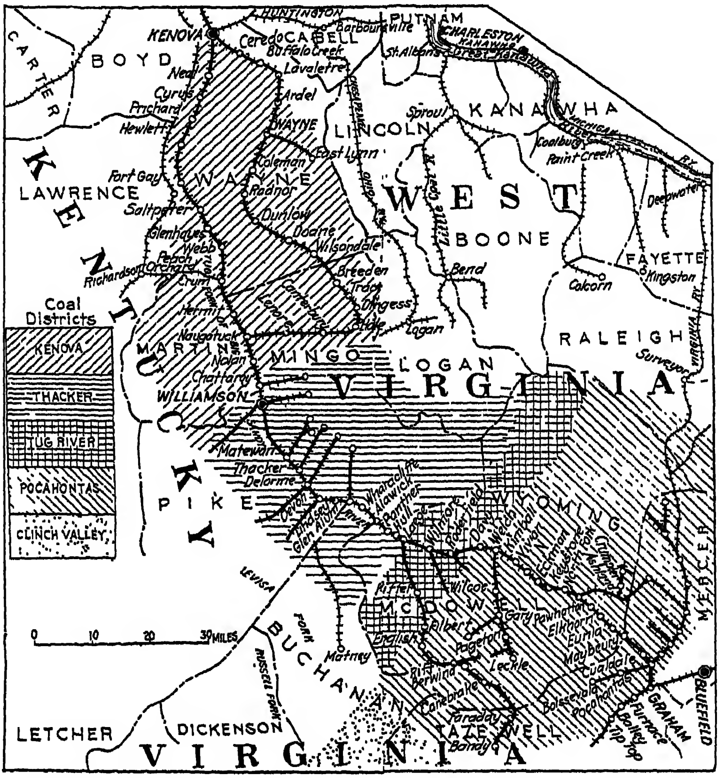

FIG. 3.—MAP OF THE NORFOLK & WESTERN RY. AND ADJACENT COAL FIELDS.

are in the bottom of the basin and were the first put down. The present field is comparatively small; it is bounded by a fault on the southeast and by the edge of the basin on the east, north, and west. As to whether the present field is only a small part of the original deposit, we can only surmise. It may have extended much farther south, into that portion of Virginia now covered by the upturned underlying measures. The coal-bearing strata of the Pocahontas field are so thick that the source of supply must have been of great extent, and carried to the Pocahontas basin by large streams in large quantities, presumably from a continent

of "Appalachia," now missing, which was located largely in what is now the Atlantic seaboard.

I. C. White gives the thickness of the Pottsville conglomerate in the Pocahontas district as approximately 3800 ft., as compared with 250 or 350 ft. at the northern border of West Virginia. This being the case, it is not inappropriate to suggest that had the geologist who named the Pottsville conglomerate had knowledge of the thickness of these measures in the Pocahontas district, they would have been named the "Pocahontas conglomerate" in preference to the "Pottsville conglomerate," as the Pocahontas district undoubtedly represents the type locality of these measures. In fact, the Pocahontas district represents the type locality of all stratified rocks in the Appalachian geosynclinal.

All the coals in the Pocahontas field are of low sulfur, ash, and phosphorus content. It is seldom that an analysis shows an ash content greater than 4 per cent.; sulfur greater than 6 per cent.; or phosphorus greater than 0.008 per cent., except where parts of the overlying or underlying strata, or parts of the bands have been included in the samples.

General Features Influencing Methods of Mining

The topography of the Pocahontas coal field is exceptionally steep and rugged, consisting chiefly of high narrow ridges and V-shaped valleys, the latter affording limited space for railroads, highways, and town sites. Considerable forethought is required in the planning of mining towns and plants. The fall of the main stream, Tug River, is approximately 0.5 per cent., except near its source. Railroads have been built along it and some of its branches with grades varying up to 3 per cent. The climate is mild, and hard winters are of rare occurrence. The rainfall is unusually heavy, from 50 to 55 in. per year, and furnishes an abundance of water for all purposes except during the four or five dry months, when artesian wells must be depended upon, as the streams become low and more or less polluted. This exceptionally heavy rainfall creates a serious problem in mine drainage.

The mountains were originally covered with abundant timber, but now all building material and about 80 per cent. of the mine timber is brought in from other localities, generally from Georgia, Alabama and Mississippi.

A large percentage of the labor now employed comes from the surrounding states, and from foreign countries. The clerical, engineering, skilled labor, and directing forces consist principally of native Americans; while the largest part of the manual labor is performed by the colored and foreign employees. The labor is well trained and efficient, diligent and

loyal. There are no labor unions in the field, although there are various secret and benevolent orders among all classes of labor.

All the mines of the Pocahontas field are served by the Norfolk & Western Ry.; the coal is easily accessible and outcrops on the mountain side throughout the greater part of the territory.

Methods of Exploration, Sampling and Estimating

Previous to the entrance of the U. S. Coal and Coke Co. into the field, prospecting, drilling and correlating of the coal seams was done in a very superficial manner. The operating companies were small and their holdings consisted of leases 3000 acres, or less, in area. The land-holding companies found no necessity for prospecting up to that time.

Development of the Tug River section was begun by the U. S. Coal and Coke Co., and railroads and towns were partly built before the property was thoroughly prospected. The Pocahontas No. 3 seam had been thoroughly developed along the Elkhorn River by operating mines, and it was supposed that this seam continued through the mountain to the Tug River side. Development of the seam outcropping on this side began in several places, but it was terminated abruptly by pinching out of the coal in every opening, at a distance of 50 to 800 ft. from the outcrop. As railroad and town building had been pushed vigorously, considerable expense was incurred by the time the coal had disappeared; as a result, the property was now most thoroughly prospected by outcrop openings and diamond-drill borings. Openings were started every 1000 ft. along the entire outcrop of No. 3 seam, a distance of 105 mi., and openings were also made on other seams. Diamond-drill cores were taken at desirable points, ranging from 1000 ft. to 5 mi. apart. This work developed the fact that the openings on the Tug River side of the mountain were not in the famous No. 3 Pocahontas seam, but on the No. 4 seam, 65 ft. above it. It further developed that under a considerable area both the No. 3. and No. 4 seams were wanting, but that, on the other hand, these two seams were in place, of good minable thickness, over a large portion of the property, and that the total area carrying both seams exceeded by about 20,000 acres the amount expected at the beginning. Had the property been only partially prospected at the start, disclosing that No. 3 seam was wanting over a large portion of the territory, and that the outcrop openings were on No. 4 instead of No. 3 seam, the lease would probably have been abandoned and development would have been retarded for years.

In the calculation of available tonnage, a line 80 ft. inside the outcrop was taken as the limit of merchantable coal, which has been confirmed by an actual average of 77 ft. Yield was estimated at 1500 tons per acre-foot, which also has proved a fairly accurate assumption.

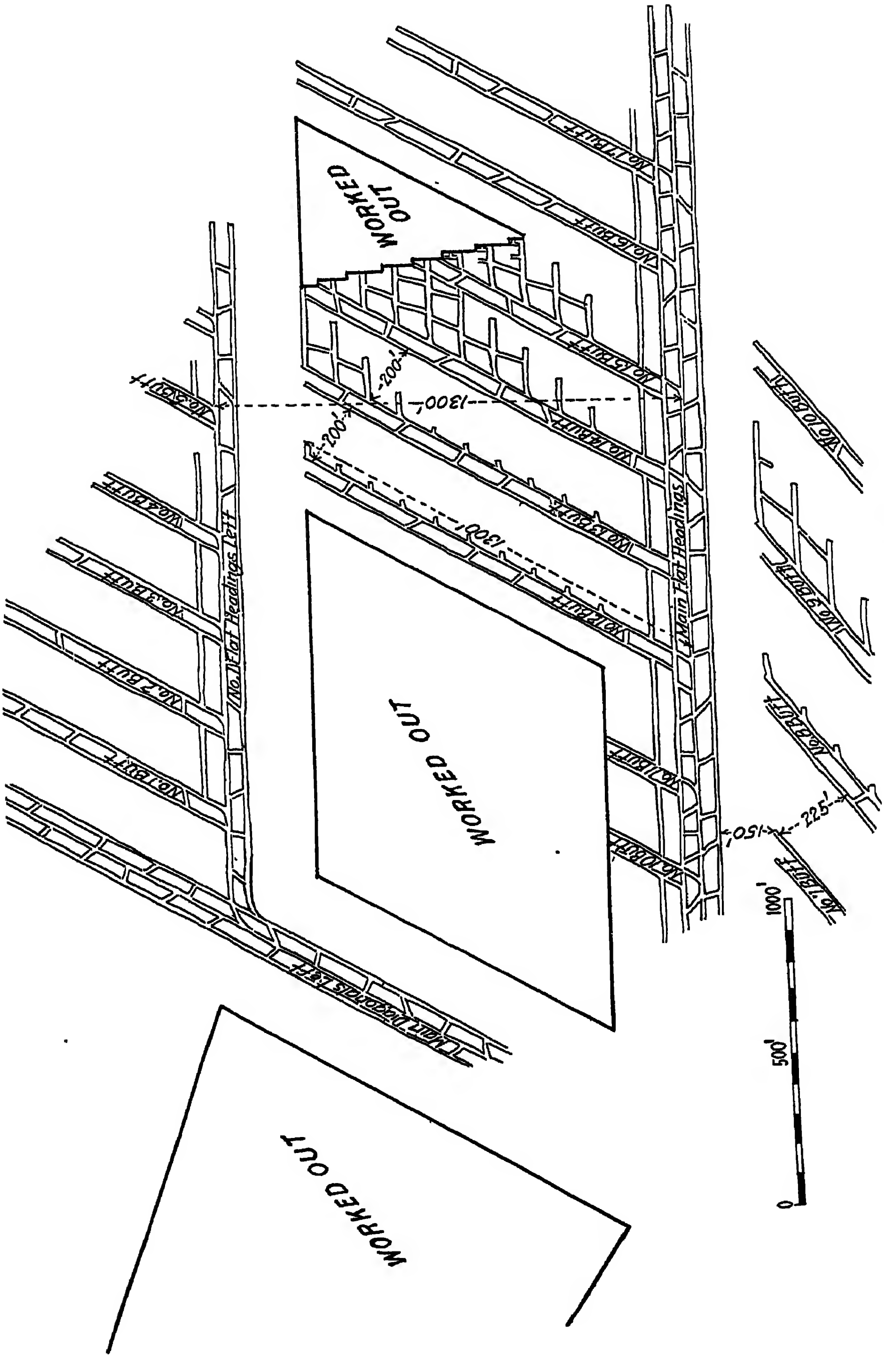

FIG. 4.—OLD PANEL SYSTEM.

Principal Former Mining Methods

The U. S. Coal and Coke Co., a subsidiary of the U. S. Steel Corporation, is the largest coal-producing company in West Virginia; its operations consist of one shaft, one slope and 12 drift openings. The shaft, which is 180 ft. deep and penetrates both No. 3 and No. 4 Pocahontas seams, has not been in operation since 1909, and is now used as a drainage outlet for the northern portion of the property.

The panel system was the first method adopted for the development of the mines on this property. These panels were above 1000 ft. square, and were developed with a set of main headings, comprising one haulage road and two airways. The intention was to drive the room headings from these main headings to the end of the panel before any room work or rib extraction was started. This method was soon found defective in the following principal respects: (*a*) Too long a time to develop; (*b*) short life; (*c*) expense of mine roads, side tracks, etc.; (*d*) expense of ventilating; (*e*) expensive to operate principal haulage; (*f*) excessive amount of coal left as barrier pillars; (*g*) loss of large percentage of this coal when removing it; (*h*) large amount of territory remaining open for a given tonnage; (*i*) large amount of equipment required for a given tonnage; (*j*) excessive cost of proper supervision, due to scattered workings; (*k*) dangerous conditions, for the above reasons.

From a portion of mine No. 6, originally developed by the old panel system in 1904, ribs and pillars of rooms that were driven at that time are still being extracted, and it will require not less than six years more to exhaust this part of the mine. Fig. 3 shows a plan of the old system, now discarded by this company.

Principal Current Mining Methods

The principal methods of mining in the Pocahontas field were described in 1915 by William H. Grady. Fig. 5 is one of a large number of illustrations in Mr. Grady's paper; it shows the details in the exploitation of two individual panels of different shapes. These systems, with slight modifications, are still in use throughout the field.

The Pocahontas No. 3 and No. 4 are the fourth and fifth seams from the bottom of the series, the interval between them being 65 to 70 ft. A typical section of each seam is shown in Fig. 5, that of No. 3 being as it occurs in Mine No. 10, while the section of No. 4 seam was measured in Mine No. 6; these mines are about 1½ mile from Gary, in different valleys.

The slate overlying No. 3 seam thins out in a northeasterly direction until it practically disappears; while the slate overlying No. 4 seam

[1] Cost Factors in Coal Production. *Trans.* (1916) **51**, 138.

thickens in a southeasterly direction until it reaches a maximum of 5 ft., before it begins to thin, and it entirely disappears in the extreme southwestern extent of the seam. The roof of No. 4 seam varies from a sand-

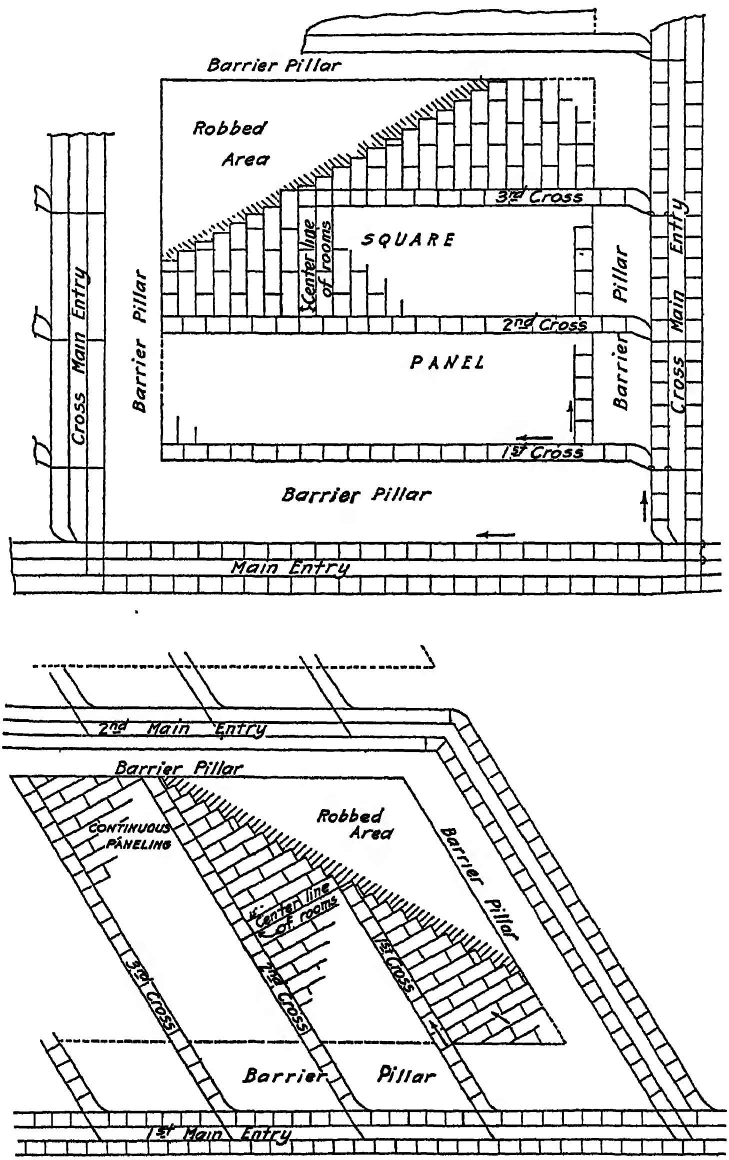

FIG. 5.—TWO TYPICAL PLANS FOR WORKING A PANEL.

stone, lying directly over the coal, to a laminated coal and slate (known locally as "rash"), which gradually appears between this sandstone and the coal until it has reached a thickness of 5 ft.; it is then gradually

displaced in a southwesterly direction, from top down, by a slate which attains a thickness of 6 ft., when it in turn gradually disappears. Where slate or black "rash" is over the coal, it renders the mining conditions difficult and dangerous.

No unusual ventilating problems have developed in the mines of the U. S. Coal and Coke Co., due to the presence of explosive gas; however, in those parts of the seam under heavy cover, explosive gas is present in small quantity, which increases as the mine workings advance. The seams dip northwesterly from 2 to 8 per cent.

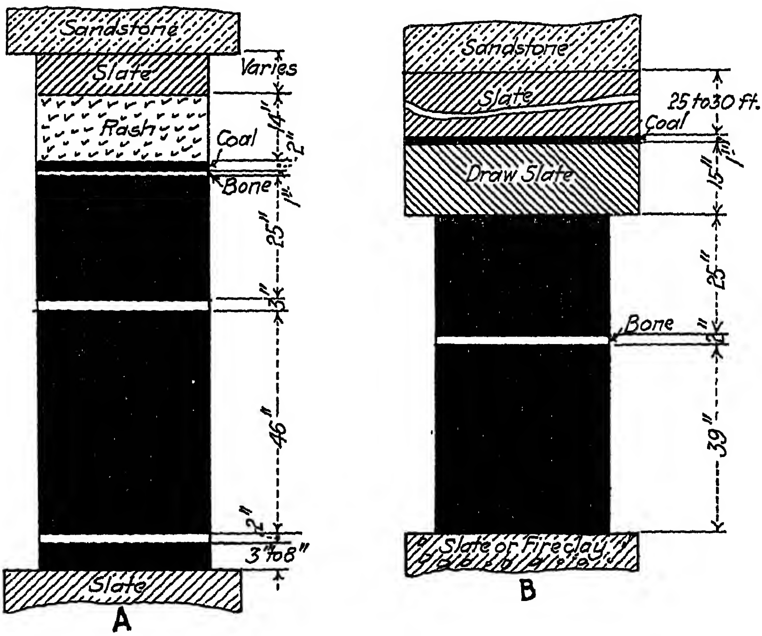

FIG. 6.—CROSS-SECTIONS OF POCAHONTAS SEAMS: *A*, NO. 3 SEAM; *B*, NO. 4 SEAM; AS DEVELOPED NEAR GARY, W. VA.

MINE OPENINGS

About 97 per cent. of the coal in the Pocahontas region is mined through drifts, on the mountain sides at various distances above the stream. The entrances are protected in various ways, such as timbering, stone side-walls and I-shaped cross beams, stone side-walls and stone arches, brick arches, or concrete arches.

From the drift openings, coal is delivered to the railroad by electric trams, steam trams, gravity planes, aerial tramways, retarding and scraper conveyors, pan conveyors, and chutes.

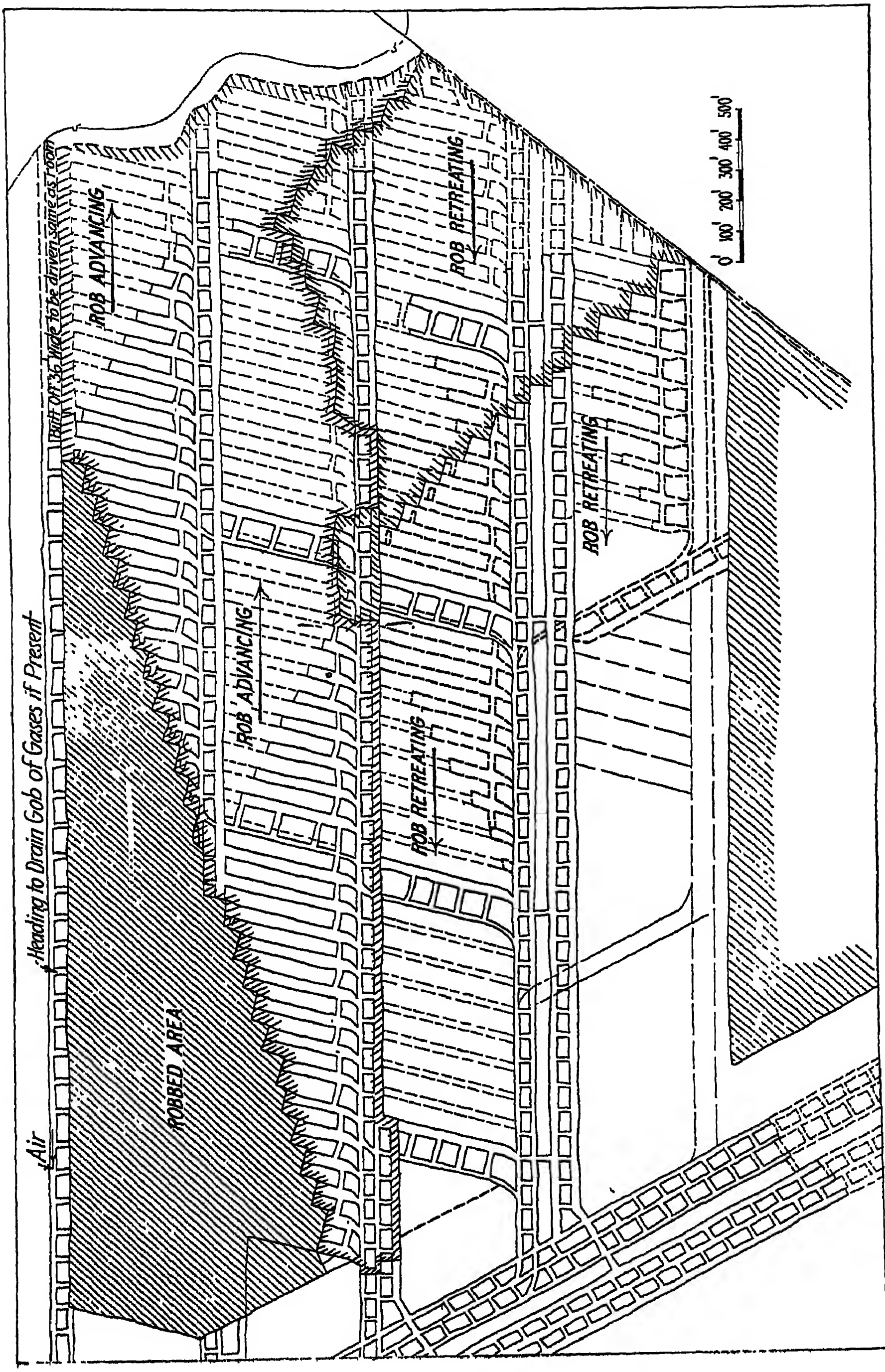

FIG. 7.—GENERAL PLAN OF OPERATION, MINE NO. 6, U. S. COAL AND COKE CO.

Underground Development Plans

The description of development plans will be confined to the practice followed by the U. S. Coal and Coke Co. at its mine No. 6 operating in the No. 4 Pocahontas seam, this plant being fairly typical of mining practice in the Gary district.

The general plan (Fig. 7) is a room-and-pillar continuous advancing and retreating system. While advancing, room ribs, heading stumps, and heading chain pillars are extracted. Thus the full production of the mine can be constantly maintained from a short period after beginning until the mine is practically exhausted. Each section is planned for a minimum capacity of 1000 tons per shift, both while advancing and retreating; this output is being exceeded in practice.

During the advancing period, the headings are continually progressing and rooms are being driven on the inbye end of them. As soon as these rooms reach their limits, the pillars are withdrawn, resulting in complete extraction over that portion of the section allotted to the advance. By this plan, the room headings last throughout the life of the mine, and become as long as the property; some such headings are now over 6400 ft. long and have two miles further to go before reaching the boundary. The life of one of these sections may be from 10 to 40 years.

This method affords simplicity of ventilation, transportation, and drainage; concentrates the workings, and permits a maximum of superintendence. No standing pillars, with resultant open work to be ventilated, are left for later difficult and expensive extraction. It is never more than two years from the time a room is started until it has reached its limit and its ribs have been robbed back to the heading.

Electric Haulage and Power

On all haulage roads, 0000 grooved trolley wire is used, and both rails are bonded. Feeder lines of 1,000,000 and 500,000 circular mill area, paralleling the trolley wire, to which the latter is connected every 200 ft., are carried on the same hanger blocks. (See Figs. 8 and 9.)

The power from a central power plant is distributed to substations, one of which is at the mine entrance, and the other 9000 ft. inside. These substations are designed to run in parallel and furnish an even distribution of power at the working face and along the haulage roads. Direct current of 275 volts is used for all underground power, including lighting, haulage and undercutting.

Electric lights are installed every 100 ft. on all haulage roads, and every 75 ft. on curves. In rooms, two or more electric lights are maintained at the face, and one light at the room switch on the heading (see Fig. 10).

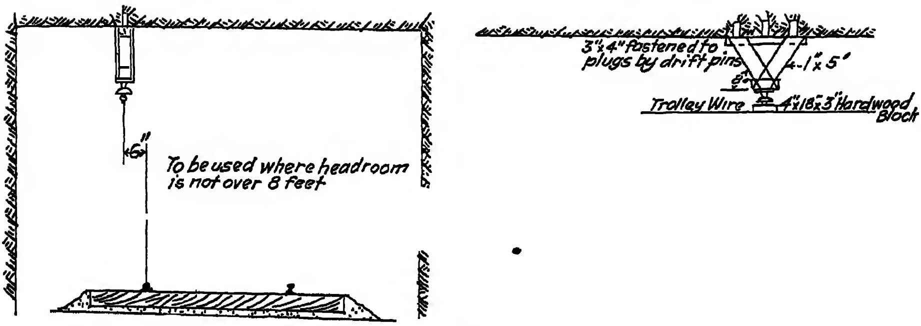

FIG. 8.—STANDARD TROLLEY SUSPENSION FOR 8-FT. HEADING.

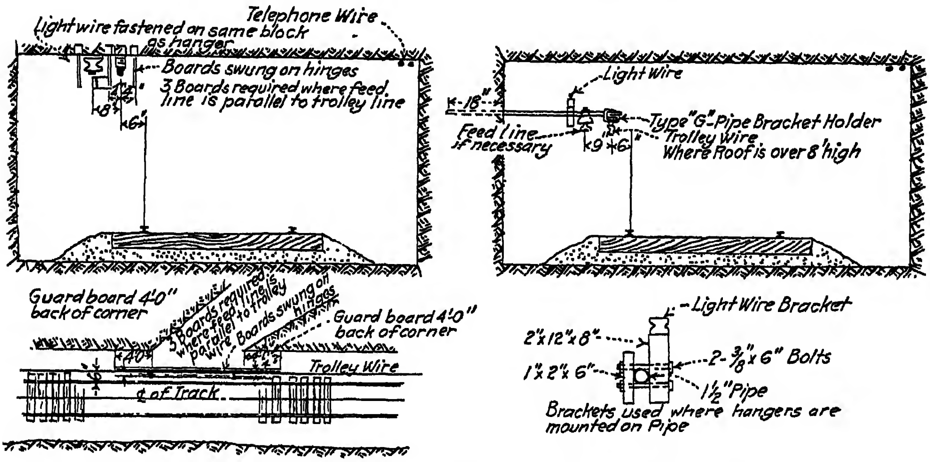

FIG. 9.—STANDARD WIRING SUSPENSION AND PROTECTION.

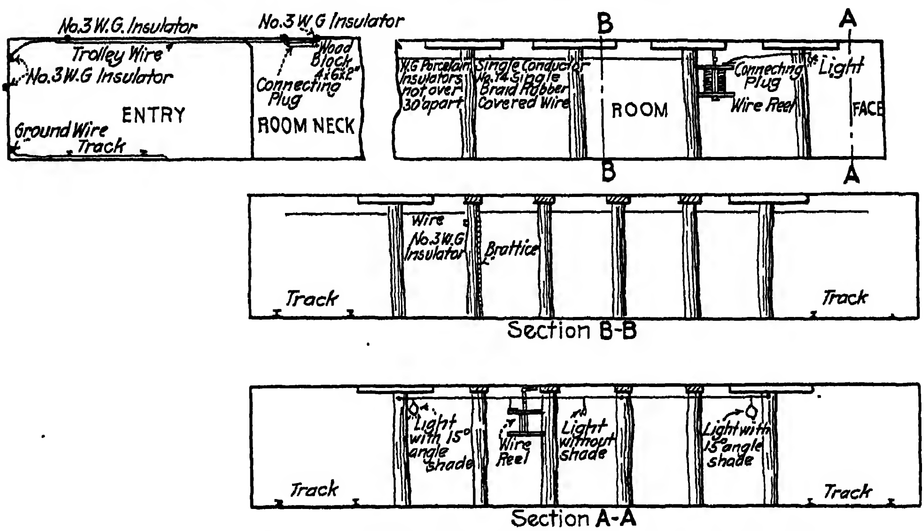

FIG. 10.—STANDARD POWER AND LIGHTING CONNECTION IN ROOM.

Drainage

During robbing, the measures generally break to the surface, and as there is no alluvial deposit or surface soil on the tops or sides of the mountains, the cracks remain open and a large quantity of water enters the worked-out areas, especially at certain seasons of the year.

Drainage is accomplished by a tunnel driven in the coal seam as near the low side of the property as possible, to which all water from this and adjacent, higher mines is drained. This tunnel is part of a drainage system extending through other mines from the lowest outcrop point on Tug River. During the period after heavy rains, weir measurements showed 12,000,000 gal. of water passing through the drainage tunnel in 24 hr. This water came chiefly from robbed areas.

Undercutting, Drilling, and Blasting

All undercutting is done with electrically driven shortwall mining machines, which are equipped with cutter bars varying from 6½ to 12 ft. long. Room and heading coal and also rib coal, when solid, is undercut. The coal in stumps and pillars, which have sufficient weight upon them to crush the coal, is removed by hand. The cover on the working places in mine No. 6 at present is so heavy that it is necessary to undercut only 42.5 per cent. of the coal mined. Table 1 shows the work done by all mining machines in the company's properties:

Table 1.—*Performance Data of Mining Machines*

Mine No.	Days Worked	Places Cut	Machines in Service	Places Cut per Machine per Day	Tons Machine Coal per Machine per Day
2	241	9,512	7	5.6	193
3	248	11,759	7	6.8	103
4	245	6,751	7	3.9	114
5	234	3,291	3	4.7	175
6	239	5,960	9	3.2	129
7	240	13,217	9	6.1	185
8	235	7,426	4	7.9	195
9	242	13,925	12	4.8	140
10	227	10,275	11	3.9	82
11	247	19,534	10	7.9	136
12	240	7,019	4	7.3	250

In the above tabulation, "Machines in Service" means the total number of machines fit for duty. The figures under "Tons per Machine per Day" are based on the total number of machines at each mine, and do not show the actual individual performance because the plants have more mining machines than necessary, these having been provided for the old panel system of mining.

With machine mining, there is a tendency to break up the coal when blasting in narrow places, while in wide places the size of the product is the same as that of hand-mined coal. In places 36 ft. wide or more, and sometimes in places as narrow as 11 ft., where the cover exceeds 600 ft., the weight of the mountain breaks the coal down from the face while being undercut.

Holes for blasting coal are of 1¾ in. diameter and are drilled by hand, with one-man twist augers. Their average depth is nine-tenths the depth of the cut. The miner drills and charges his holes with a permissible explosive and electric detonator, and tamps them with clay brought from the outside. The charge is fired by the assistant mine foreman during the day shift, with a pocket battery which he carries for that purpose.

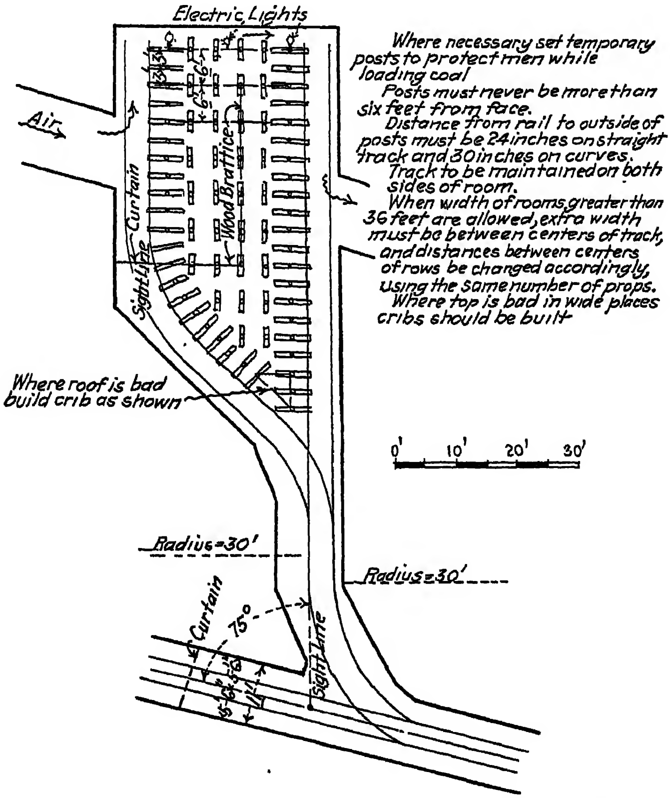

FIG. 11.—STANDARD TIMBERING FOR 2-TRACK ROOM.

ROOMS AND PILLARS

Rooms are turned on a schedule which permits pillar extraction to commence immediately upon the completion of the room (Fig. 7). Rooms are spaced on 125, 90, and 75-ft. centers, and are driven 60, 36, and 24 ft. wide, according to thickness of cover and roof conditions.

Only one track is required in places 18 ft. or less in width. Track is laid next to the outbye side of the room in advancing sections, and is aligned by sights. Places 36 to 40 ft. wide have two tracks, one on each side of the room, and aligned by sights. In rooms 60 ft. wide, three tracks are used, one in the center of the room and one on each side. With rooms 36 ft. or more in width only one crosscut is driven between rooms, this being 80 ft. from the heading; ventilation is maintained from this point by a wooden brattice in the center of the room. (See Fig. 11.)

Partings in the seam are picked out by the miners, and gobbed. The amount of bone thus removed by the miner is approximately 5 per cent. of the seam. The coal is soft, has no definite cleavage planes, and breaks with a columnar fracture. There are three bone partings in No. 4 seam, one being a few inches above the floor, another in the middle of the seam, and one at 2 or 3 in. from the top. The bottom parting and the coal below it, which is only a few inches thick, are not mined. The middle and top partings are mined with the seam and partially removed by the miner before loading; but the 10 to 14 in. of overlying "rash," being soft, breaks into small particles, mixes with the coal, and is loaded with it.

Timbering

No timbering is permitted on haulage roads without written permission of the general superintendent, which is given only in exceptional cases. Where the roof is bad, it must be taken down until all loose material is removed. Concrete posts are used on main headings where permanent support is required.

Timbering in rooms follows a standard plan (Fig. 11). Timbers having a minimum diameter of 5½ in. are used. A row of "road posts" is set parallel to the room track, 2 ft. from straight track or 30 in. from curved track, and 3 ft. apart. A cap piece 6 ft. long, placed on the top of each road post, extends over the track. The other timbers, "gob posts," are spaced 6 ft. apart and use a cap piece 4 ft. long. Timbers are maintained at a distance not greater than 6 ft. from the working face, irrespective of condition of roof, and closer where necessary.

No preservatives are used and there is no general scheme for recovering timber, only a small portion of which is reclaimed. The timber is delivered to the working places by locomotives and mules during the day shift, and placed by the miner as required. Table 2 shows the amount of timber per ton and per acre used for all purposes, including mine car repairs, ties for mine roads, caps, props, and miscellaneous:

TABLE 2.—*Timber Consumed, Board Feet*

Mine No.	Average Thickness of Coal, Ft.	Bd. Ft. per Net Ton	Bd. Ft. per Acre
2	5.75	2.8	32,164
3	4 53	5.6	48,600
4	6.32	4.0	66,075
5	6.94	3.4	40,318
6	6.18	1.4	16,263
7	6 37	3.5	45,453
8	5 97	4.2	43,750
9	7.36	2.3	26,611
10	5.52	5.2	47,685
11	5.35	4.4	45,411
12	8.04	2.6	33,083
Average.........	6.06	3.3	38,163

HAULAGE SYSTEM

Mine tracks and hauling equipment have been standardized. A gage of 48 in. is standard for straight track, 48½ in. on haulage curves, and 49 in. on room curves. The standard radii of curves and switches are 100, 150, 200 and 300 ft. for roads, and 30 ft. for room turnouts. Rails are 60-lb. on main haulage roads, 40-lb. on room headings, and 20-lb. in rooms and temporary headings. The main haulage road is double tracked and has no grade in excess of 2.5 per cent. Before any permanent track is placed on the main or branch haulage roads, a profile is made and it is graded according to elevations.

Room track is placed and maintained by the miners; all turns and switches are placed by company men. Ties 5½ in. × 6½ ft. are used under 60- and 40-lb. steel rails; ties 4½ × 5 in. × 6 ft. are used under 20-lb. steel rails, where conditions do not warrant the use of steel ties. On haulage roads, ties are spaced not over 24 in. and steel ties for temporary track are spaced not less than 36 in. Permanent track in headings is maintained not less than 150 ft. from the face.

On haulage roads, the minimum clearance is 2½ ft. on both sides, from any part of a locomotive or car; and the least vertical distance is 5½ ft. above the top of the rail. Where grades exceeding 2 per cent. are encountered, the floor or roof is excavated to allow as favorable a grade as possible. Room headings are on an average grade of 0.5 per cent. in favor of the loads.

No side tracks, or partings, are required by the system of transportation in use. Mules are used only for hauling cars into and out of the working places, and hence can serve a larger number of places, with less work, than under a system involving side tracks.

The locomotive enters the room heading, pulling a trip of empty cars, which are distributed at the mouths of the various working places by disconnecting them from the rear end of the trip; at the same time, it pushes together the loaded cars, placed on the heading by the mules, until its trip is assembled. The motor then passes through a haulageway to a parallel heading and thence in the direction of the tipple; these haulageways from the main heading to the room heading are driven at intervals of 800 ft. (see Fig. 7).

Usually a 13-ton electric locomotive will gather from the mouths of the working places, and deliver to the outside, 600 to 700 tons of coal per day, when the round trip does not exceed 4 miles. From five to seven haulers, and seven to ten mules, depending upon the grades in the working places, are required to gather the coal hauled by one locomotive.

Mine Cars

Wooden mine cars, of 93 cu. ft. capacity, are used. The axles are of 3 in. diameter and run in plain bearings. The wheels are cast iron, of 18 in. diameter, and run loose on the axle. The wheel base is 48 in., the same as the gage.

In 1921, each car made an average of 1.7 trips per shift; at that time, 508 mine cars were in service at this plant. At present, the average is over two trips per shift. In 1919, each car traveled an average of 878.4 miles loaded, and made an average of 507.7 round trips, the average haul at that time being 1.723 miles each way; it thus performed 3039 ton-miles of work. In 1921, the cost of mine car maintenance, per ton, was $0.0422 of which amount $0.0179 was for labor and $0.0243 for material.

Locomotives

Four 13-ton electric locomotives and one 26-ton tandem locomotive are used for all gathering and main haulage work. The four smaller locomotives frequently handle as much as 875 tons apiece; their average for 1921 was 612 tons each, per shift. The tandem locomotive hauls from 1662 to 1750 tons per shift, with a maximum haul of 12,000 ft. The most advanced workings are 17,000 ft. from the tipple.

Labor Efficiency, Wages, Etc.

The company employs no so-called contractors; miners, and machine runners are paid on a piece basis, and loaders on a car basis (Table 6). Day labor is paid by the hour (Table 5).

Table 3 gives the output, in tons per man-hour, based on records for May and June, 1922, when 1,027,795 short tons were produced.

TABLE 3.—*Labor Efficiency*

	Tons per Man-hour	Man-hours per Ton
Miners in headings, incl. machine runners. (Best miners usually assigned to headings.)	2.115	0.463
Miners in rooms, and at pillars, incl. machine runners	1.870	0.534
Miners on slate or rock work (est.)	2.500	0.400
All miners	1.910	0.523
All underground day labor	1.850	0.540
Men on transportation system	5.570	0.179
All surface labor, excl. office force	3.380	0.295
Total organization, excl. general office and engineers	0.728	1.365

TABLE 4.—*Classification of Day Labor*

	Per Cent.		Per Cent.
Inside Labor:		*Outside Labor:*	
Mine foremen	0.60	Clerks	1.26
Fire boss	0.16	Janitors	0.09
Asst. mine foremen	6.13	Stablemen	0.94
Roadmen	9.27	Teamsters	2.06
Day labor	22.70	Garbage men	0.45
Bratticemen	1.29	Laborers	4.85
Wiremen	1.36	Labor foremen	0 48
Pumpers	0.78	Pumpers	0.77
Drillmen	0.19	Electricians	0.93
Machine repairmen	0.22	Motor repairmen	1.36
		Substation men	1 65
	42.70	Blacksmiths & helpers	1.75
Transportation:		Car repairers	4.63
Haulers (mule)	13.81	Carpenters	3.58
Motormen	3.47	Tipplemen	9.83
Brakemen	3.33	Greasers	0 59
Trappers	0.53	Plumbers	0.19
		Masons	0.18
	21.14	Machinists	0.17
		Watchmen	0.05
Total inside day labor	63.84	Supply clerks	0.13
		Painters	0.15
		Sanitary inspectors	0.07
		Total outside door labor	36.16

While no satisfactory records are kept from which the percentage of labor turnover can be figured, it is believed to be small, as compared with other mining communities. A considerable number of the employees have been working for the company 15 to 18 years. The superintendents employ all men, and discharge them for defective work. Mine foremen have authority, under the state mining law, to discharge for infraction of the mining laws.

Table 5 gives the wage scale, in force at the close of 1922, for miscellaneous day labor, mostly on the outside; their shift is 9 hours.

TABLE 5.—*Scale of Wages for Day Labor; Late* 1922

Occupation	Per Hour	Occupation	Per Hour
Stableman (per month)	$145.00	Laborers, inside	$0.70
Teamsters	0.60	Haulers, inside; mule	0.80
Carters	0.55	Motormen	0.80
Mason tenders	0.55	Boss drivers	0.95
Outside laborers	0.40	Motor brakemen	0.84
Chargers	0.75	Loaders under tipple	0.70
Charger trailers	0.60	Dumpers	0.70
Car shifters	0.55	Car catchers	0.65
Drafters	0.55	Car pushers	0.65
Oven waterers	0.55	Car droppers	0.70
Coke machine boss	0.65	Pumpers, inside	0.75
Outside pumper & pipe man	0.65	Checkman	0.65
Watchman	0.55	Tipple foreman	0.94
Substation man	0.60	Greasers	0.60
Crusher repairman	0.65	Picking-table men	0.55
Blacksmith	0.90	Slate dumper	0.55
Slate picker	0.55	Bratticeman	0.80
Assistant mine foreman	0.95	Wireman	0.80
Shop foreman	0.90	Motor repairman	0.85
Machinists	0.90	Mining-machine repair man	0.85
Carpenters	0.80	Masons	0.80
Car repairman	0.90	Machinery inspector	0.85
Roadmen	0.89	Firemen	0.75

Table 6 gives the coal-mining and loading prices in force at the close of 1922. Miners in this occupation usually work about 8 hr. a day.

TABLE 6.—*Coal-mining and Loading Rates; Late* 1922

Machine Rates	
Room up to 24 ft., with 8-ft. cutter bar, or less:	
Runner	$1.70 per place
Helper	1.30 per place
Room over 24 ft., with 8-ft. cutter bar, or less:	
Runner	$1.90 per place
Helper	1.55 per place
Room up to 24 ft., with cutter bar over 8 ft.:	
Runner	$1.80 per place
Helper	1.50 per place
Room over 24 ft., with cutter bar over 8 ft.:	
Runner	$2.30 per place
Helper	1.85 per place
Room over 45 ft., with cutter bar over 8 ft.:	
Runner	$2.45 per place
Helper	2.00 per place

Coal-loading Rates, per Car	
Room	$2.00
Heading	2.12
Wet heading	2.15
Mining & loading by hand (pick)	2.10
Slate	1.80
Black rash	1.15

Power Consumption and Cost

Table 7 gives the consumption of power during May and June, 1922, when the total production was 1,027,795 short tons, the figures including

all transmission, transformer, and conversion losses. Table 8 distributes the cost of power, per ton mined.

Table 7.—*Power Consumed, Per Ton and Total*

Item	Kw.-hr. per Ton	Total Kw.-hr.
Cutting	0.25	256,949
Hauling	0.77	791,402
Pumping (inside)	0.138	141,836
Pumping (outside)	0.613	630,038
Ventilating	1.372	1,410,135
Tipple machinery	0.225	231,254
Lighting (outside)	0.3	308,339
Lighting (inside)	0.075	77,085
Shop & miscl	0.2044	210,037
Total	3.9474	4,057,075

Table 8.—*Distribution of Power Cost, per Ton Mined*

Mining	$0.00250
Hauling & hoisting	0.00995
Pumping	0.00138
Ventilation	0.01372
Shop & miscl	0.01192

Consumption of Material

Table 9 distributes the cost of materials and supplies not previously included in the cost of power; the percentages are based on the total cost of these materials, not on the total cost of production.

Table 9.—*Distribution of Cost of Materials*

Item	Per Cent.	Item	Per Cent
Mine roads	18.66	Cleaning, crushing & washing coal	0.25
Mine cars	8.45	Fire protection	0.06
Haulage system	14.65	Other machinery repairs	0.50
Hoisting & dumping	1.00	Safety	0.06
Shaft, tipple & bins	0.31	Welfare	0.13
Ventilation	3.57	Miscellaneous	4.83
Drainage	1.31	Electric lights	2.88
Machine mining	4.45	Pit posts, caps & cross-bars (3.3 bd. ft. per ton)	24.49
Explosives (0.117 lb. per ton)	1.31		
Horseback, rock & refuse	0.25		
Live stock	10.65		
Lubricants	2.19		100.00

Safety and Welfare

Safety is the first consideration in all plans of organization, mining, construction, installation, and operation. Specifications for all machinery must call for proper guards before quotations are requested. A standardized system of mining was devised as the best means of reducing the hazard of the work; the standard clearance on all headings and

standard timbering in all rooms, as well as the removal of all slate in the headings, are among the most efficient provisions that have been adopted. Premiums are paid monthly to underground foremen for good accident records.

Frame dwellings are provided, all of which have electric lights in every room. Artesian wells have been drilled to supply water for all purposes, and the water is filtered. Running water is piped into all houses, and a drain system is provided. A number of the dwellings have baths and other modern conveniences.

Concrete and macadam streets and concrete sidewalks are laid on the principal streets, with concrete walks to the front and back doors of a great many of the houses. The Americans, both white and colored, and the foreign employees live together in harmony in these towns; the only segregation requirement is that white and colored shall not live in the same house.

Churches for white, colored, and foreign employees, are situated at the company's plants.

Separate schools, managed by the district board of education, are provided for white and colored employees; these are rated as "First Class" by the State Department of Free Schools. The smaller children are taught at local schools, close to their homes, while Junior and Senior High Schools are centralized at five different points on the plants of the company; the larger children are carried to school in automobiles during the entire school year. Gary High School is an accredited entrance school for the University of West Virginia, and several other universities of equal standing.

Amusements are provided by moving-picture theaters, bowling alleys, pool rooms, baseball grounds, tennis courts, and the various athletic clubs. All employees can participate in these amusements. Club houses are provided among the various plants of the company, in which board and lodging are provided for the single employees, and such married people as do not care to keep house. These clubhouses are also used for community gatherings. The recreation of the younger people is generally conducted under the auspices of the church or school authorities.

Provisions, household necessities, etc., are supplied through an affiliated concern, the United Supply Co., which conducts a mercantile business at eight different points, in and around Gary. Each store has seven departments. As the stores purchase their goods in large quantities for cash, and their business is conducted on practically a cash basis, their prices to the consumer, after allowing for a fair profit, are lower than those of merchants in surrounding towns, such as Welch, Wilcoe, or Anawalt.

Ground is rented by the U. S. Coal and Coke Co. to outside parties for running stores, devoted principally to the coarser grades of merchan-

dise, such as feed, flour, etc., delivering to the purchasers by trucks. All employees are paid in cash, and they are under no obligation to trade at any particular store.

A nurse has been employed by the board of education and the local chapter of the American Red Cross. This nurse visits the schools and examines the children, particularily for physical defects, but also for cases of malnutrition, and other matters affecting their health. A dental clinic is provided by the county. The dentists visit each school. examine the children's teeth and do such dental work as is necessary; this is all without charge to the parents.

Acknowledgments

In conclusion, the writer wishes to thank W. C. Stratton, chief engineer of the U. S. Coal and Coke Co., N. D. Maher, president of the Norfolk & Western Ry. Co., C. S. Churchill, vice-president of the Norfolk & Western Ry. Co., W. F. Hanley, division engineer of the U. S. Coal and Coke Co., and Guy C. Mace, assistant to the general superintendent of the U. S. Coal and Coke Co., for information and assistance in the preparation of this paper.

DISCUSSION

S. A. Taylor, Pittsburgh, Pa.—About what tonnage of coal corresponds with the performance data of the drainage ditch mentioned on page 888?

Edward O'Toole.—This tunnel starts in the coal seam at the lowest point of outcrop. It is driven in a northwesterly direction with a grade of 1.22 per cent., while the coal seam rises in a southeasterly direction at 6 or 7 per cent., until at the apex of the anticline it is over 550 ft. above the drainage of the valley and 800 ft. above the drainage tunnel, at a point 32,100 ft. from the outlet. The mines drained through this tunnel are Nos. 2, 6, 7, and 9, which have a total capacity of 16,000 tons per day, and most of the water from these mines drains through this tunnel.

We have kept no regular record of flow, except during periods of high water, when 12,000,000 gal. per 24 hr. passed through it. We started the tunnel 18 years ago, and have had the use of it in a limited way ever since; its usefulness is increasing with its age. We had to enlarge portions of it three or four years ago, as the volume of water at times was greater than it would readily carry at those places. At present, we do not care very much what amount of rainfall we have as it runs through the mine and out through the tunnel and we have no further concern with it. I do not believe the cost of maintaining the tunnel exceeds $200 a year.

S. A. Taylor.—The purpose of my question was not to determine the cost, but to get an idea as to the relative volumes of water and coal. In the Pittsburgh district we have found, by a number of measurements and tests, that there are 7.5 tons of water handled for every ton of coal. In the anthracite region the amount is much greater than that; I was told recently that it was 11.5 to 12 tons, per ton of coal, but I have seen statements that it was almost double that amount. It was my impression that in your district the mines yield 5 tons of water for every ton of coal produced.

James E. Jones, Switchback, W. Va.—The amount of water drained in 1922 from the mines mentioned, as measured by a recording weir, was 904,871,000 gal. The tonnage from the area drained was 1,080,062 tons, or 3.11 tons of water per ton of coal mined.

Edward O'Toole.—When the water is held in the upper measures of the broken strata for a considerable time it becomes acid. There appears to be a stratum in the upper measures that contains sulfur and the water when passing through this slowly absorbs quite a quantity of it; but when the amount of water is large and passes through these measures rapidly the acid is not noticeable.

INDEX

[NOTE.—In this Index the names of authors of papers are printed in SMALL CAPITALS, and the titles of papers in *italics*.]